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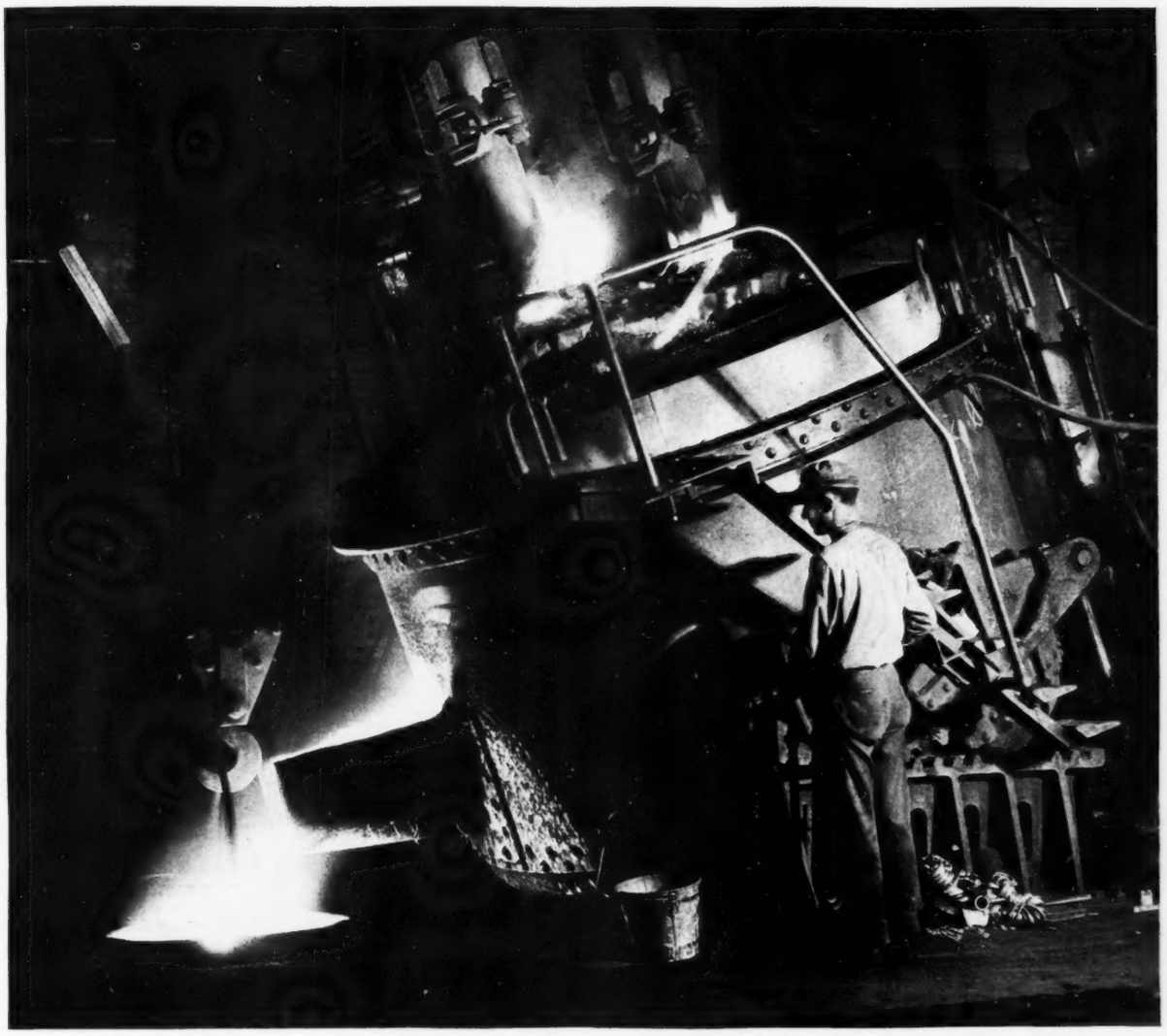

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November, 1932

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November, 1932 — Engineering & Mining World — Vol. 133—No. 11

A. W. ALLEN, *Editor*

Adjusting Employment To New Conditions

INDUSTRY, in all the more progressive countries of the world, is facing many serious problems, chief among which is that caused by excess of labor supply over labor demand. The increasing use of machinery, particularly in recent years, has created idleness; and general economic conditions everywhere operate against the possibility of early readjustment by the creation of new wants, stimulation in the manufacture and sale of the so-called luxury products, and by making available and popular new services.

By a process of eliminating other less-feasible plans, attention is being focused on the opportunity for benefit in the adoption of shorter working hours. Over two years ago this publication, in discussing the problem, recommended attention to a plan whereby, for ordinary industrial operations, uninterrupted production could be secured during six full week days, with Sunday reserved. Two days off per week were to be allowed, but the week-day vacations would be staggered from Monday to Saturday, enabling the employee to have one week day off, in addition to Sunday. Continuous operation over six days would thereby result, instead of only five or five and one-half days. More workers would be employed. Shopping during week days by employees would be spread over six days, instead of being concentrated in one afternoon. Provisions for amusement and recreation would be utilized every day, instead of only on one and a half days. The growing congestion of the roads in and around centers of population, especially on Sundays, would be considerably reduced.

No broad recommendation can be made that will meet the peculiar requirements of each operating company, but the impression is too prevalent that any program for the staggering of employment, any plan that will by such means help to reduce the number of the jobless, must adversely affect production output and operating costs. Such a contention is disproved by the results of a change made by the Kellogg Company, in Battle Creek, Michigan, that involved reduction in the hours of work from eight to six, by an increase to four shifts per twenty-four hours. Although introduced primarily as a contribution to the problem of unemployment, the change has redounded to the benefit of the company. The plan necessitated a reduction in day-wage rates for higher-paid employees, but an increase was found possible in hourly base rates. A greater output per hour resulted, the waste of time involved by the necessity for lunch periods was avoided, a larger return was made on capital investment, and the overhead expense per unit was proportionately reduced. The plan has been in operation for nearly two years, and will doubtless be adopted as permanent policy.

Serious thinkers are united in their conviction that the profound changes effected by the rapid introduction of the machine in recent years can be prevented from permanently disturbing economic equilibrium only by a drastic and general reduction in the hours of labor of each employee. Most mining companies, unfortunately, are not in a position to contribute to national well-being by any change in employment practices. Other companies are more fortunately placed. To them the suggestion is made that abandonment of the eight-hour shift in favor of the six-hour shift might raise the efficiency of operations without increasing the unit cost of production. It would certainly effect a desirable addition to the number of men employed, and so contribute to industrial rehabilitation.

National buying power, which controls general business activity, is dependent to a major extent on the income of wage earners. One sure avenue of economic recovery lies in an increase in the number of employees, appreciation in hourly wage rates as a result of improved individual efficiency, a greater demand for commodities and for services, a demand that will automatically stimulate activity in capital expenditures for equipment, and for the raw materials used by manufacturers.

Planned Research and Industrial Control

RESEARCH endeavor in the mineral industries has expanded rapidly in recent years, not only in scope and number of investigators but also in the extent to which varied agencies have entered on such activity. So rapid and widespread has this expansion been, in fact, that to keep informed concerning all possible sources of new knowledge in any particular field has been difficult, if not impossible. At the moment, owing to the general economic situation, research is receiving marked attention, and to it industry must look for major aid in seeking a way out of current difficulties. In this connection a recent publication of the U. S. Bureau of Mines, "Research Activities in the Mineral Industries of the United States," is of timely interest. Questionnaires inquiring into the problems engaging their attention were sent to industrial, institutional, university, and governmental research departments. A mimeographed catalog of 285 pages is the product of the survey, and even a cursory examination of its contents is enlightening. For instance, under the heading of beneficiation, in the section dealing with metallic ores and products, the Ford Motor Company is revealed as investigating the beneficiation of low-grade specular iron ore; a manufacturer of vapor lamps is

studying the use of ultra-violet light to produce fluorescence and phosphorescence in minerals as an indicator of the degree of separation effected during beneficiation; an equipment company is endeavoring to adapt its concentrating appliances to the recovery of metals from foundry residue and other waste materials.

The future of the mining industry is associated intimately with the search into the unknown that the catalog records. Cognizance of these forces is advisable, if for no other reason than to aid in correlating such effort, which objective apparently prompted the survey. But, if a planned industry in the years ahead is desirable, planned research is an essential step in reaching that goal. Herein rests, perhaps, the most significant aspect in which this contribution of the Bureau's may be considered.

Wise Counsel For the Learner

THE address of the incoming president of the Institution of Mining & Metallurgy, Dr. Sydney W. Smith, delivered at the annual meeting of that society in London, was replete with sound philosophy and helpful advice; but the pedant will demur at his statement that "It is an old saying that a little knowledge is dangerous." But, he added in contradiction of that dictum, "unless one endeavors steadily and persistently to acquire the little knowledge, which is often undeservedly despised, the imagination is never fed with the material which will enable it to function actively—subject, of course, to due restraint. I would say to all students of mining and metallurgy," enjoins this distinguished assayer, "lose no opportunity of picking up information regarding the principles and technique of any other branch of technical activity which comes within your ken."

This is excellent counsel, particularly in these trying times. Of major importance in the building of a balanced intellectual structure is diversity of interest and knowledge. And the knowledge may be acquired—indeed, it is usually acquired—in small doses. The alleged contempt of a "little knowledge" to which Doctor Smith refers is caused by repeated misquotation. The warning originally uttered by Alexander Pope was as follows:

> "A little learning is a dangerous thing:
> Drink deep, or taste not the Pierian spring."

Huxley, a hundred years later, in "Science and Culture," discussed the possible outcome "If a little knowledge is dangerous . . . ," and so stimulated the misconception that is now so common, arising from the thoughtless substitution of "knowledge" for "learning."

As Doctor Smith so sanely advocates, excursion in search of knowledge should be stimulated. A deep understanding of the meaning of life and of the significance of its phenomena is possible only by a steady absorption and digestion of facts, mostly of insignificant individual weight. With education or learning, however, care must be taken to avoid the hazards incidental to the acquisition of merely a veneer of information on any one branch of human endeavor. Retention by the memory of a smattering of facts, without appreciation of the fundamental principles involved, is no evidence of true education or intellectual attainment.

Reducing the Incidence To Tuberculosis

MOST of those who work in mines are exposed to an insidious hazard to health, and no advantage can be gained by ignoring the fact that tuberculosis, acquired underground or in surface plants where dust is not allayed, is a cause of preventable disability and mortality. Vigorous combative and educational measures are being taken by the leading mining companies in all parts of the world, with excellent results; but the parish of *Engineering and Mining Journal* includes a multitude of properties where the risk is recognized but where no vigorous corrective action is being taken. The "skeleton in the cupboard" is ignored, simple truths are not recognized, and the result is misery, suffering, and financial loss to all concerned.

Generally speaking, however, the ravages on humanity of no other ailment have been reduced by the persistent dissemination of educative propaganda to such an extent as has been the case with tuberculosis. Research and medical science have combined to simplify diagnosis and treatment, rendering the disease amenable to care and attention. Notwithstanding this fact, the individual whose health and happiness and earning power are threatened usually fails to heed the warnings of approaching trouble. Early symptoms in most instances are unmistakable, but often indiscernible to close friends and relatives—an inexplicable fatigue, loss of body weight, and a persistent though perhaps insignificant cough. Cure is almost certain if prompt precautions are taken; and no remedial action is too drastic if this can be effected.

In its campaign against phthisis the National Tuberculosis Association of the United States has met with signal success, as statistics indicate. But it cannot seek out incipient invalids. To them and to executives who have the health of their employees at heart this publication suggests that advantage be taken of the excellent advice contained in circulars and pamphlets obtainable on request from the Association's headquarters at 450 Seventh Avenue, New York City.

As the Christmas season approaches, the plea will be made to use Association seals on envelopes carrying cards and letters of greetings. Funds are needed, and a large response should be assured. To help in the suppression of tuberculosis, particularly in the mining industries, is more than a duty.

Roan Antelope Mining Practice

R. M. Peterson

OF THE various properties developed in the copper belt of Northern Rhodesia, Roan Antelope is the first underground mine to be brought into production. Detailed descriptions of the district and its development have appeared at various times in engineering periodicals. A. Chester Beatty described its early history and prospecting work in the December, 1931, issue of *Mining and Metallurgy*. R. J. Parker and Anton Gray detailed its geology in the September, 1929, issues of the *Engineering and Mining Journal*. A general description of the Roan Antelope mine, mill and smelter, by A. G. McGregor, appeared in the *Mining Magazine* (London) of November, 1930. The object of this paper is to set forth some of the mining problems encountered at the Roan Antelope property, to indicate how those problems were solved, and to describe the methods used to bring underground operations into low-cost production.

As the Roan Antelope was the first underground mine in the district (Bwana M'Kubwa being an open pit), the first step taken was to ascertain the reliability of drilling results. To accomplish this, several incline shafts were sunk to various depths. West of the Luanshya stream, No. 1 shaft was sunk to a vertical depth of 320 ft., and drives were advanced east and west at a vertical depth of 280 ft. East of the Luanshya stream, No. 2 and No. 3 shafts had lateral openings at inclined distances of 430 and 630 ft., which correspond to the present 280-ft. and 420-ft. vertical levels. Along these drives the formation was crosscut every 200 ft. Sampling of these crosscuts furnished the data to prove up the continuity of the orebody and the reliability of the drill records in regard to tonnage and grade of the ore. These crosscuts were also available to any examining engineers who cared to check the results. On the basis of this work, confidence was established, and to date no further exploratory work has been done underground to the west. Drilling results now indicate 108,000,000 tons of ore averaging 3.44 per cent copper, of which 95 per cent is in the form of sulphides, with a large area underlain by the ore series as yet unexplored.

Chalcocite is the predominant sulphide at the east end of the orebody, with bornite and chalcopyrite predominating in the westerly extensions. The minerals are finely disseminated throughout a 10- to 45-ft. band of well-bedded metamorphosed argillaceous sandstone and shale, the bedding planes of which are quite pronounced and exert considerable influence on the choice of stoping methods. The hanging wall shades into lean and barren shales, quartzites, sandstones, and dolomites. On the footwall of the ore shales is a narrow band (generally about 3 ft. thick in the present stoping areas) of calcareous biotite schist, which in some places has a high copper content. Below the schist are feldspathic quartzites and conglomerates. West of No. 2 shaft, the mineralization transgresses from the schist into the shales, and forms a shale footwall for mining purposes. Oxidation has left little sulphide copper above the 120 vertical level. The upper limit of the stoping operations is determined by the amount of sulphides in this transition zone of combined oxide and sulphide values. The ore is remarkably uniform in grade.

Structurally, the eastern part of the orebody is an asymmetrical plunging syncline, its axis striking about N. 65 deg. W., its east end rising to the surface so that the outcrop is "hair-pin" shaped. Mining operations have started at this eastern end, which, for a length of about 16,000 ft., has been described as being shaped like a canoe, with the north side bent in at the top. The greatest width of this "canoe" is about 2,000 ft., and its greatest vertical depth is about 2,200 ft. In spite of the comparatively sharp bend at its "nose," the formations contain no major fault, a fact which makes complicated mining plans unnecessary in this area.

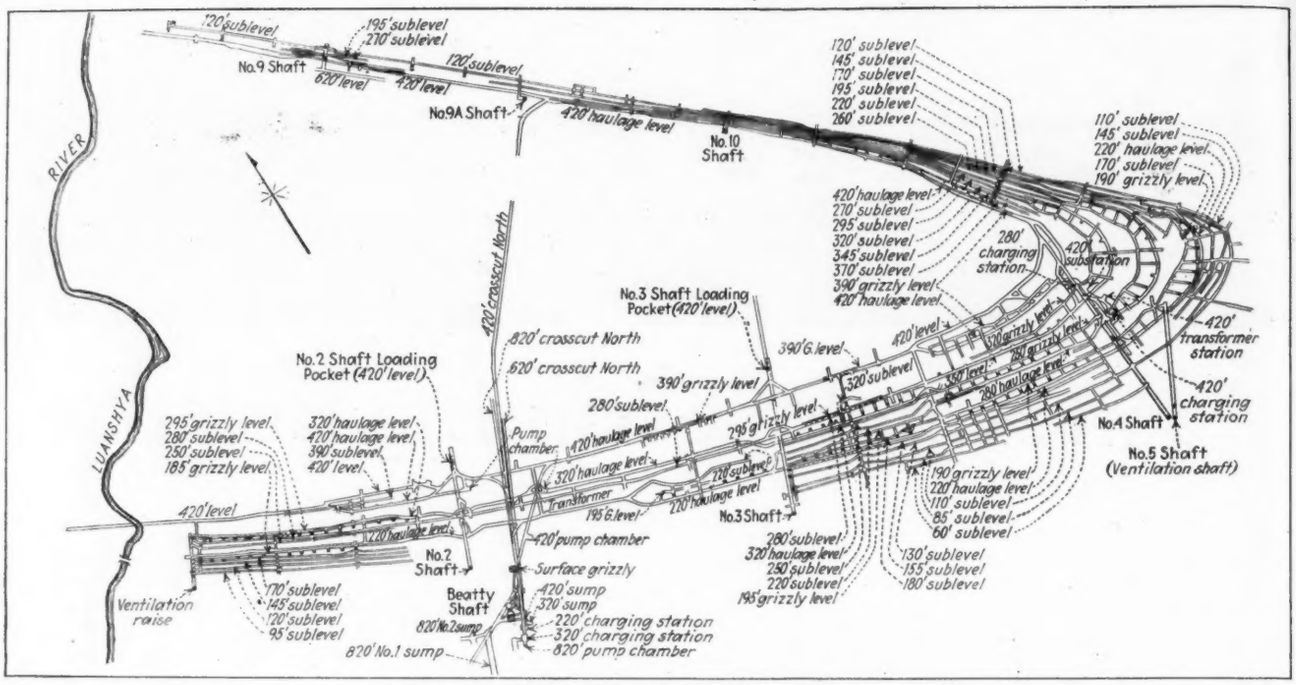
As the Luanshya stream cuts at right angles across the orebody, 2,700 ft. from the east end of the outcrop, a pillar of ground is being left beneath it. The first stopes are between this pillar and the "nose."

Geographical and topographical considerations determined the location of the township and surface plants. The township is situated on a low hill within comfortable walking distance of the surface plants, and consists of modern bungalow type homes equipped with electric lights, and a waterborne sewage system. Excellent drinking water is pumped 8 miles from the Kafubu River.

A well-equipped hospital was erected, and is maintained by the company. The mine recreation club is complete with library, reading room, electric gramophone with loud-speakers, bar, and soda fountain. Tennis courts, soccer and rugby fields, and a nine-hole golf course are maintained by the various athletic organizations.

Much money has been spent by the company to eradicate malaria from the camp, and a "control area" of more than twelve square miles has been drained and treated to prevent mosquitoes from breeding near Luanshya. These efforts were highly successful.

The successive steps taken in deciding on permanent mining plans were: (1) location of permanent shafts or shaft; (2) method of stoping; (3) location of main levels; (4) execution of development plans with special regard to existing openings and equipment. After weighing the advantages and disadvantages of several preliminary schemes, a decision was made that the logical solution was one main vertical shaft for the eastern portion of the property, located 180 ft. back in the footwall of the south outcropping, and 1,500 ft. east of the Luanshya stream. This shaft delivers ore directly to the coarse-crushing plant, thereby eliminating all surface transportation.



Plan of underground workings, Roan Antelope mine

A drill hole was put down to test the formation while detailed plans were being worked out. The first 46 ft. of Beatty shaft, including the concrete collar, and all sinking preparations were completed in August, 1929, when active sinking began. Rate of progress was influenced by the quality of the native labor that could be procured to lash or muck out the broken rock. Lashing in shaft sinking is hard work, and the natives did not have the endurance to stand up to it. The rate of progress in sinking the shaft depended entirely on getting the round lashed out. No great breaking problem existed, and enough water to impede the work was never encountered. A 10-B Cameron vertical sinking pump handled the flow of water.

A day's work consisted of one breaking shift and three lashing shifts. Installation of permanent steel sets was kept within 50 ft. of the bottom. The 220, 320, 420, 620, and 820 stations, also the skip-loading pocket excavations below the 420 and 820 levels, were cut as a part of the sinking program. Maximum progress in one month was 98 ft. In September, 1930, the shaft was finished at a depth of 985 ft. Details of the shaft are presented on page 25 of the January, 1932, issue of *Engineering and Mining Journal*.

An ore-pass system permits ore to be dumped near Beatty shaft at every level, from surface to the 820. The ore is hoisted from the skip-loading pockets below the 420 and 820 levels. A large finger-type control gate, with the fingers constructed of native hardwood lined with manganese steel, is in the ore-pass system at the 620 level. The ore pass itself is about 6x8 ft. in section, but pockets 20 ft. in diameter, at the 420- and 820-level stations, where the Granby cars are dumped, have capacities of nearly a thousand tons each. Therefore, in the complete ore-pass system at Beatty shaft, including pockets, a storage capacity of about 4,000 tons is provided.

While shaft equipping operations were being completed, the permanent steel headgear was constructed over the temporary wooden headgear. Work was carried on simultaneously, so that little delay was occasioned by the change-over. The ore bin is emptied by means of two Ross chain feeders onto two 60-in. belts, which feed two 30-in. gyratory crushers. From these, the ore is carried by belt conveyors to the Symons crushing bins of the mill system. While Beatty shaft was being sunk and equipped, and the hoisting apparatus installed, underground experience was gained, so that initial stoping plans could be determined. Also development was carried on at top speed.

A small trial stope had been opened up on the north limb above the 120 level, and from information gained there, general conclusions were drawn as to the probable action of hanging wall and footwall during future stoping operations. Several mining schemes were formulated, and from them the main development was laid out. Existing openings were put to the best possible use. To have haulages fairly close together to begin with was considered best, so that any possible error in judgment would be on the conservative side.

Haulageways in ore 220, 320, 420, 620, and 820 ft. vertically below the surface were decided upon for the Roan side. On the Rietbok side a decision was made to mine the ore from the oxide zone to the 420 level in one operation. In the flat section at the "nose," where the ore flattens out on the pitching bottom of the syncline, haulageways in the footwall were planned at the 220, 280, 350, and 420 elevations. The haulages in the "nose" deliver ore through ore passes to the 420 level.

While underground work was in the development stage, surface construction work was scheduled, so that plant units became ready in the order of their need. Offices, shops, stores, and warehouses went up first, closely followed by crushing plants and concentrator units.

The steel-sharpening shop is on the surface near Beatty shaft. Drill steel is loaded from racks, into cars, which are pushed over to the shaft and onto the cage, and then taken to whatever level is desired, where they are pushed in to the points from which the drill-steel is distributed.

No rehandling of drill steel occurs between the shop and main level destination. Special drill-steel cars have been made by building box bodies on standard Granby wheels and underframes. Each of these cars has capacity for seven tons of steel. Six sharpeners and two tempering furnaces handle sharpening and hardening requirements. A well-equipped rock-drill repair shop has facilities for precision grinding of pistons, cylinders, and valve parts.

Pumping is at present from temporary stations on the 420 and 820 levels. Under construction on the 820 level is a pump station and a 1,000,000-gal. sump. Half of the sump is for clear water. The other half, for muddy water, is divided into two parts by a

reinforced-concrete wall, so that one part may be cleaned out while the other part is in use. The sump is 350 ft. long, 20 ft. wide, 20 ft. deep at the ends, and 26 ft. deep at the middle. Provision is made for a duplicate sump that can be excavated easily in the future, if required. Dirty water will run into a launder, from which six quieting boxes project halfway to the bottom of the sumps. Clear water will overflow into level launders running the length of the dirty-water sumps and will empty into the clear-water sump, from which a 16-in. suction pipe will carry it to the 1,000-g.p.m. centrifugal pumps.

Three of these pumps constitute the initial installation, but space is being provided for two more, if found necessary. The pumps are designed to deliver against a total head of 870 ft., running with five stages at a speed of 1,450 r.p.m. They are coupled to 400-hp. motors supplied with 3-phase, 50-cycle, 3,300-volt current. As the pump room is below the level of the sumps, thus providing a 27-ft. head on the suction side when sumps are full, a drainage sump has been provided to collect leakage and gland seal water. Watertight doors isolate the pump room, so that in the event of a sudden large flow, the pumps will not be drowned out before water fills the shaft bottom, all 820-level workings, the sump, and Beatty shaft up to a point 32 ft. above the 820 station, where the top of the pipe raise is situated. Two 200-g.p.m. Glenfield and Kennedy sludge pumps will deliver sump sediment and pump room leakage into a separate 6-in. pipe line to the surface.

Sludge pumps are vertical, three-throw, ram type, with rams 7-in. diameter and 10-in. stroke. They are designed to deliver water containing 30 per cent solids against a head of 950 ft. Valves and seats are of cupro-nickel, with balata rings and phosphor bronze springs. Driving power is furnished by 125-hp., 50-cycle, 3-phase, 3,300-volt motors. The pump chamber is connected to Beatty shaft by a 45-deg. pipe raise, through which electric cables, 16-in. water and 6-in. sludge mains are carried, and then up Beatty shaft to the surface.

After the preliminary layout of underground openings had been decided on an intensive development program was planned. Methods of driving which combined economy with the maximum of speed also had to be developed.

With raw "boys," many of whom had come direct from the bush and had never before seen a European, much less a mine, some means of mechanical mucking had to be developed. To hope that laborers who did not know how to use shovels could muck out a heading in any reasonable period of time was of course vain. Remarkable advances have been made by the native during the past three years. This not only is the result of his own ability to learn but also reflects the "teaching" policy of the company.

The first black workmen were painfully crude. To them, a shovel was a novelty, to be used only when the "baas" made them stop scooping up the dirt with their hands. A pipe wrench was an intricate mechanism entirely beyond comprehension. One of the first problems was to get them sufficiently strengthened for a full day's work. Owing to undernourishment, few were able really to accomplish much in an eight-hour shift. After subsisting precariously on "mealie meal" (ground corn) all his life, the black laborer was hardly in physical condition for arduous toil.

Another problem was the big "turnover" prevalent in the early days. "Boys" would work a few months, and then return to the "kraal" and buy another wife. One of the most prominent characteristics of the native is his capacity to do well those jobs that require the tiresome (to a European) repetition of tasks. In some ways this characteristic is a handicap, but in many mining operations it appears to be a desirable quality. A constant aim of the European bosses is to arrange work so that advantage may be taken of the characteristic, as the inevitable result is what every one is striving for—namely, lower costs.

Early in 1930, scrapers were introduced in headings, and a portable slide was developed that fits the conditions, and has proved an unqualified success. It meets the specifications of low first cost, high salvage value, cheap repairs, high mucking speed, and, most important of all, is readily operated by native labor. In fact, the natives take delight in operating scraping equipment, and have reached a high degree of efficiency in this work.

The best drifting record to date was made in April, 1932, when a 10x12-ft. drive in ore was advanced 277 ft. in 25 drilling and 25 lashing shifts; this is an average of 11.1 ft. per round. The scraper had no difficulty in cleaning out the drift in an eight-hour night shift. The crew was standard for this type of development, and consisted of one European miner in charge of seven "boys" who ran two 4-in. drifters, with 13-ft., 1¼-in., hollow-round, lugged steel. Lashing was in charge of one European. A battery locomotive and Granby cars did the tramming, and a native scraper operator was helped by six other natives in "bossing up" the scraper and switching.

Drilling in a 5x7-ft. sublevel heading is done by a gang of three natives with a light jackhammer. A European miner easily "bosses up" three or four headings. Drilling records of three miners for the month of October, 1930, are presented in Table I.

Table I—Drilling Records in Subdrifts and Subraises

	Miner No. 1	Miner No. 2	Miner No. 3
Total length of subdrifts and subraises driven, ft.	764	607	565
Advance per 50 lb. case of 60 per cent powder, ft.	6.4	6.5	7.3
Advance per machine shift, ft.	5.8	6.0	5.3
Advance per native shift, ft.	1.8	1.9	1.5

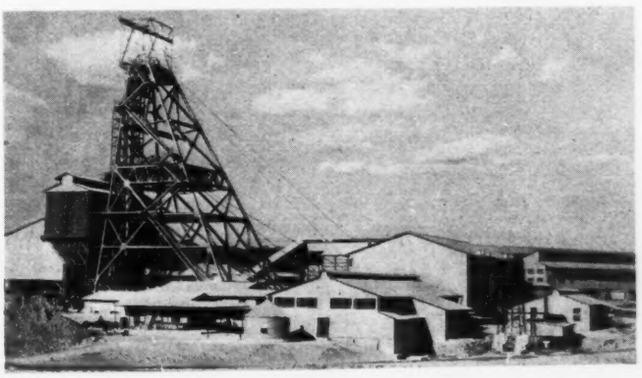
Miner No. 1 did his own pipe work and laid 18-in. gage, 18-lb. track in the 480 ft. of sublevels that he drove.

Lashing in a sublevel is done by a gang of four natives, with shovels and wheelbarrows or small U-type side-tipping cars. Peculiarly, psychology plays an important part in this work. Up in a small heading, natives will attack a comparatively small muckpile and reduce it in reasonably good time. But double the number of natives, working twice as large a muckpile in a big heading, will muck nowhere near the same number of tons per shift. First, the sight of a huge pile induces them to "take it easy," and second, they have come to look on mucking in a large drift as being the job of "lo shteama" (scraper).

Although large and small drifting was soon on an efficient basis, a much longer time was required to train natives to become proficient in raising, and raising costs per foot were high at the start of the development program. By the end of 1931, however, the natives had "learned the knack," and raising costs sank to a commendable level.

In December, 1930, development work was at its greatest intensity, 8,115 ft. To get rid of the ore and rock from these headings, advantage was taken of the auxiliary shafts, which were also

Headframe and plant



used for ventilation and servicing. Connections were established around the east end of the orebody on the 420 level, between shafts Nos. 9, 10, 3, 2, and Beatty shaft crosscut. No. 4 shaft was sunk from the surface and raised from the 420 at the "nose." This shaft is useful as a service incline and ventilation raise, because it is needed no more for hoisting development tonnage.

Pockets and skip-loading facilities were installed on the 420 level at both No. 2 and No. 3 shafts, which were equipped with 1½-ton skips. Provision for an ore stockpile was made on the surface at No. 3 shaft, and for a waste dump at No. 2 shaft. Development tonnage on the south limb was segregated underground. Similarly, north limb rock and ore found their respective outlets at vertical shafts Nos. 9 and 10.

The first stopes, and all development and stope preparation prior to the initial stages of production, were laid out and worked according to detailed plans, which may be summarized as follows:

1. Dips over 50 deg.: Sublevel stoping; 5x7-ft. sublevels, 25 ft. apart vertically; 5x7-ft. grizzly level, 35 ft. above a 9x9-ft. haulage in ore; chutes and 10x15-ft. grizzlies on 50-ft. centers; two finger raises per grizzly.

2. Dips 40 to 50 deg.: Sublevel stoping; grizzly level 25 ft. above haulageway in ore; chutes and 7½x10-ft. grizzlies on 25-ft. centers; one stope opening per grizzly.

3. Dips under 40 deg.: inclined room-and-pillar mining, with scraping; 25-ft. room started from a central "mining raise," and stoping from bottom to top conducted from a horizontal trail and bench; grizzly level 31 ft. above 9x9-ft. haulageway in footwall; chutes and 7½x10-ft. grizzlies on 50-ft. centers.

A short description of the Roan Antelope sublevel method of mining open stopes may be of interest. The main haulage level, as shown in the accompanying underground plan, is driven in the ore and near the footwall side, so that the grizzly will be installed on the footwall of the ore. The grizzly level is driven 35 ft., in elevation, above the haulage level, and has short connecting crosscuts to the grizzly excavations. The first sublevel is driven about 17 ft. above the grizzly level, so as to intersect the tops of the cones formed by the raises from each end of the grizzly. Then the first sublevel is located in the most advantageous position for opening out the bottom of the stope in the form of a cone from footwall to hanging wall. The remaining sublevels to the top of the ore or stoping block are driven in the middle of the ore at 25 ft. vertical intervals. Service raises are driven at convenient intervals, depending on how many stope faces must be provided in a certain block of ground, or on the economical distance of handling drill steel and supplies to the working face.

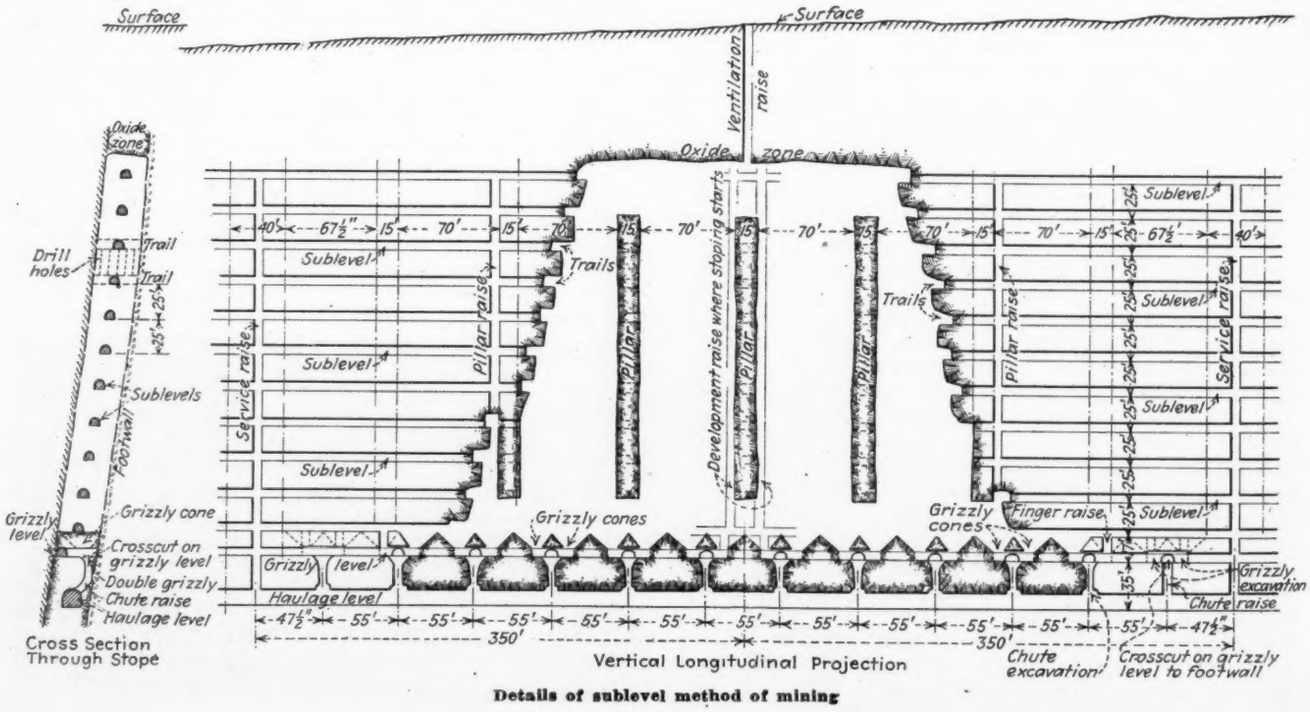
When the sublevels have been driven in a block between service raises, mining operations are started by connecting the sublevels by a raise, and then widening out this slot to the footwall and hanging wall. The raise is put up on the footwall, and short connections are made to the sublevels. After the slot has been widened to the walls, the regular sequence of mining operations takes place simultaneously in both directions.

The slot at this stage of stope development presents a face on either side from which further breaking will take place. Sublevels give access to this face at 25-ft. vertical intervals. The next operation is to drill slabbing holes from the sublevel toward each wall. When these holes are blasted, a bench or trail is formed. Workmen can stand on this trail, and drill subsequent rounds until the trail has been advanced to the hanging wall on one side of the sublevel and to the footwall in the other direction.

Holes are then drilled down with a jackhammer, and up into the back with a stoper. The up-and-down holes from any two consecutive benches or trails meet, or are close enough to break the intervening ground. The benches are then blasted singly or several at the same time. Bottom benches are advanced ahead of the top benches, the angle of the face depending largely on the strength of the ground and the operating conditions. Pillar spacing depends on the strength of the walls, structural weaknesses, and seriousness of dilution. In sublevel stoping, these conditions are under control at all times as the pillar spacing can be varied and changed to suit conditions. To leave a pillar the sublevels are connected by a raise and stoping operations started again, as from the original mining raise.

Sublevel trails are convenient for sampling control of stoping operations. The samplers have access to and drill test holes into the walls. Mineralization generally follows the bedding. From these data, information is given the bosses as to the distance from the center of the sublevel to the walls, at all the sublevels in the stope. This information is kept up to date as the work progresses.

After production was fairly under way, and experience had been gained



Details of sublevel method of mining

with the various types of ground, with reactions and capacities of European and native labor, and with the equipment used, still further modifications and improvements suggested themselves. From the experience gained, a study has been possible of the mining methods as regards the structural features of the orebody and, possibly most important, respecting their adaptation to the most efficient use of native labor. Modified to suit local conditions of ground and labor, the sublevel method is progressively proving that it is an ideal method for our conditions. The most important factors in this situation may be enumerated as follows:

1. The working place can always be barred and made safe. The sublevel exit is always at hand.

2. Working conditions are similar day by day, and drilling operations are repeated. Native labor is most proficient at repetition tasks.

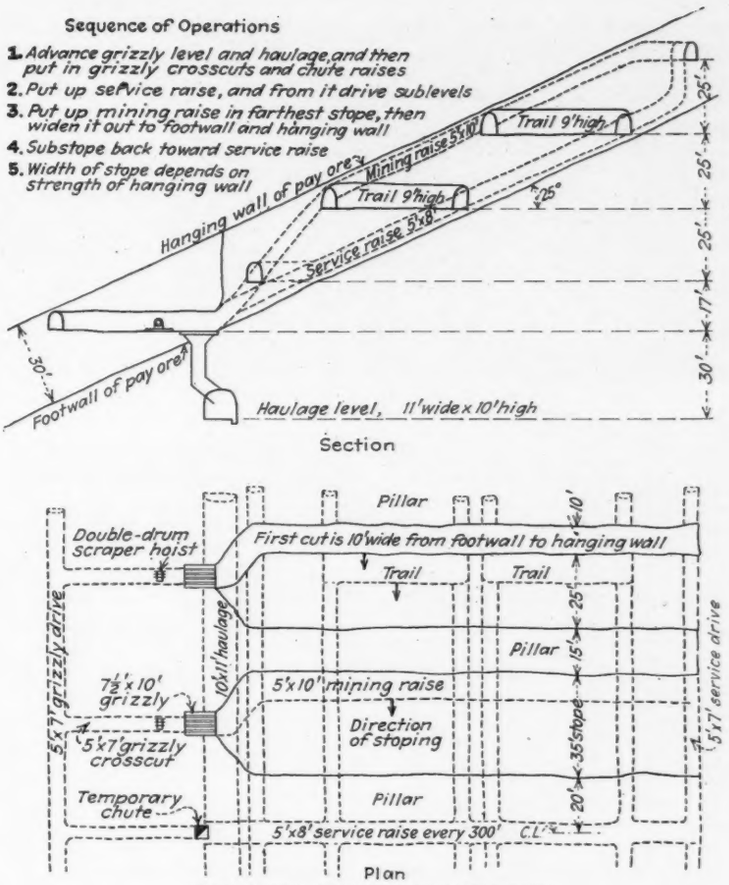
3. Working conditions are similar in all the stopes.

4. Pillar spacing can be varied at will, and as mining conditions demand.

5. Stoping operations can be shut down completely or opened up to full capacity at short notice, without affecting grade or causing wastage of reserves.

On dips greater than 40 deg., ore will slide well on the shale footwall with no assistance, but below that dip scrapers are necessary. Flat stopes, according to original plans, were to be worked from a central mining raise, the benching beginning at the lower end of the stope and progressing up the dip. Bedding planes, however, are so well defined that the ore breaks to them and leaves a slippery inclined plane on which natives cannot work safely. Also, in the flat sections, the hanging wall has not been found to be uniform, and the correct span is hard to estimate. The present system of flat stoping, embodying the principles of sublevel stoping for breaking operations, has therefore been adopted. As shown in an accompanying illustration, a "mining" raise is put up from the grizzly, and along one side of the planned stope; this is opened out from hanging wall to footwall; then the ore is substoped back along the strike to the required width, which is at least 35 ft. Then a 15-ft. pillar is left, and, on the other side of this pillar, operations are repeated. This repetition of operations and similarity to breaking methods in the steeply dipping stopes is a factor of prime importance where native labor is concerned.

As these flat parts of the mine are also places where the ore is thickest, convenience is gained by having two sublevels on the same elevation, the second sublevel being driven as a part of stoping operations. "Pillar raises" are also driven by the miner in charge of stoping "boys," and simultaneously with stoping operations. Another advantage of the method is that scraping is done easily and safely, as tail sheaves may be reached by way of the sublevel benches, which are well protected from falling ground.



Method used in mining flat-dipping ore

Future haulageways for levels below the 420 will not be driven in the footwall, but in the ore. Thus, eventually, all development will be in the ore, making unnecessary special arrangements for the handling of waste.

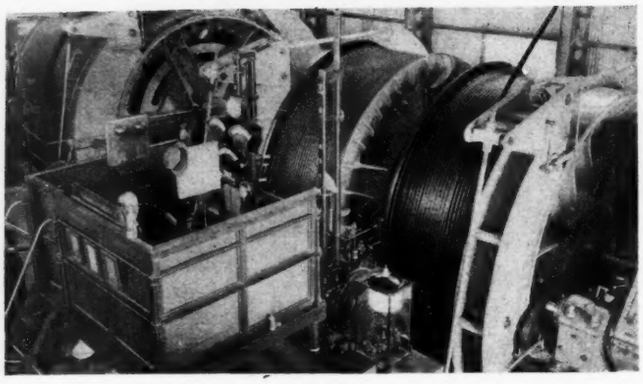
In steeply dipping stopes, the first stope openings were carried as wide as possible along the strike, to determine what span of ground could be carried without pillars, and to add to the knowledge concerning dilution by sloughing from the hanging wall. This experiment showed that, subject to local slips, the safe span left between pillars should be approximately 70 ft. Simplification of mining in the steep portion of the orebody has been accomplished by adopting the 50-ft. spacing of chutes in all sections, with the 10x15-ft. grizzly and two finger raises opening into the stope. Also, owing mainly to experience gained by white and native miners, sublevels may be at 35-ft. vertical intervals, which makes for considerable saving in development.

Three standard sizes of grizzlies are in use. For scraping stopes and places where small quantities of ore are run, a 7½x10-ft. grizzly, built of 10x4½-in. I-beams and ¾x6-in.x5-ft. manganese-steel plates, is used, with openings 16 in. wide.

Where a grizzly taps two stope openings, and large quantities of ore are run, a 10x15-ft. size is used. To the original design of these grizzlies, more sidewise bracing has been added in the form of short pieces of I-beam. The original I-beam bars have been reinforced for lateral blows by covering them with inverted channels. Owing to the relatively slight abrasive character of the ore, and to the price of manganese steel, the wearing plates have been changed to mild-steel, ¾-in. flats, 15 ft. long. Pieces of this length give added strength, besides simplifying handling and installing. This type of grizzly has handled over 450 tons in an eight-hour shift.

Machines used in stopes are jackhammers, using ⅞-in. hollow hexagon steel, and stopers, using 1-in. hollow hexagon steel, without lugs. Several gangs of two to three "boys" each are supervised by one European miner. Stoping records show that, during the last six months, 312.9 tons was mined per European shift, 18.0 tons per native shift, 79.4 tons per machine shift, and 2.89 tons per pound of powder.

Ample ventilation is provided by shafts and raises from the 420 level and by No. 8 prospect shaft. A fan with a capacity of 25,000 cu.ft. per minute was placed at the raise on the Roan



Man and ore hoists

limb to exhaust stope gases. Shafts Nos. 3, 5, and 8 are each equipped with a fan having a capacity of 50,000 cu.ft. per minute. When conditions warrant, a fan with a capacity of 100,000 cu.ft. per minute will be placed at No. 10 shaft to exhaust Rietbok limb gases. Good air is downcast at Beatty shaft, No. 2 shaft, No. 4 shaft, and the Rietbok limb ventilation raise.

During the development period, chutes were used that employed the overcut arc principle. As the rock at this stage was comparatively uniform in size and well broken, no trouble was experienced. For handling ore from stopes, a chute was installed, using the undercut arc principle. At first this chute was opened by hand, its size requiring three or four native operators. Adding an 8-in. air cylinder reduced the operating staff to one native, and also obviated the necessity for an operating platform on the hanging-wall side of the drift, which took up a lot of headroom, requiring extra excavation, and prevented headroom for trolley wire at the chutes. The cylinder is also faster and more positive in action. A feature of the altered design is an angle frame supporting three to four pieces of lagging. As the ore is flowing through the chute, these pieces move up, owing to the pressure exerted on the bottom piece by the ore stream, thus automatically providing a larger opening. When the gate is closed and the flow stops, the lagging automatically settles down and stops overflow.

In tramming from large development ends and chutes, 92-cu.ft. Granby cars are used. They have proved satisfactory; and their simple dumping, sturdy construction, and small repair costs are ideal for hauling economically large quantities of ore in trains operated by native crews. Trains of sixteen cars are drawn by 8-ton trolley locomotives, having a speed of 6.5 miles per hour at 4,000-lb. drawbar pull, when operating at 250 volts. The wheel base of the locomotives is 61 in., and the wheels are 30 in. in diameter. Two 50-hp. motors per locomotive are controlled by "dead-man" handles. Power is furnished by a motor-generator set installed on the surface in the Beatty shaft hoist house. Two 200-kw. sets constitute the installation, which also furnishes current for the surface direct-current requirements.

For gathering purposes, for short hauls, and in development ends, 6-ton storage-battery locomotives are used. Two motors per locomotive are equipped with magnetic control, to make, if desired, 12-ton tandem units. Wheels, 24 in. in diameter, are inside the frame. The wheel base is 60 in. Speed is 3.5 miles per hour at rated drawbar pull of 2,400 lb., when operating at 85 volts. Edison batteries of 80 cells are contained in a battery box equipped with rollers for moving sidewise off the frame. All locomotives and Granby cars are equipped with half-size Alliance automatic couplers.

A standard crew consists of a driver and two helpers. On the "in" trip a helper stands on a platform on the last car of the train and signals with a police whistle. At one section of the 420 haulage level, red and green hand-operated lights facilitate traffic. The "staff system" was used successfully when four trains were running. The possessor of the staff had authority to be in the block. Natives readily carried out the procedure.

Trackage consists of 45-lb. rails at 30-in. gage. Native hardwood sleepers, 5x10 in.x5 ft., are laid at intervals of 2 ft. 6 in., and are well ballasted. Care is taken to have track conditions exactly right, and ditches alongside of the tracks are kept clean. The standard turnout is a No. 4 frog, used in conjunction with a curve of 80-ft. radius; but where convenient in main haulage drives, a No. 6 frog and a curve of 180-ft. radius are installed.

Oregon pine sideboards, 2x8-in., are affixed to the brackets holding the No. 2C figure-eight trolley wire. These brackets are spaced at about 12-ft. intervals along the drives, and are held by 1½-in. pipe lengths plugged into holes drilled into the hanging. As each "boy" starts underground work, he is carefully warned about the danger of touching trolley wires, and as a result of the precautions noted in the foregoing, not one "boy" has come into contact with a live wire.

The first skips of ore heralding regular production were hoisted on May 21, 1931, and the process of tuning up the plant commenced. On June 1, production was started in earnest, and mine output steadily increased until Jan. 1, 1932. At this time, in conjunction with all companies in agreement with the world copper conference held in New York, production was curtailed. It was reduced still further on April 1, 1932.

Until the smelter was ready, concentrate was shipped. The policy of the company was not to increase production until the concentrate could be smelted on the property. An increase in the tonnage rate does not therefore represent the capacity of the mine to deliver tonnage and to reduce the costs by operating on a balanced program. The underground staff, during this period, were primarily interested in silling out stopes, and in getting the main level and grizzly work prepared for a continuous large production. The first cakes of blister copper were poured at the smelter on Oct. 20, 1931. Underground production and cost figures are presented in Table II.

Table II—Underground Production and Costs

Month	Ore Mined, Short Tons	Mining Cost, Shillings per Ton
June, 1931	35,500	14.37
July	54,500	10.14
August	79,800	7.17
September	107,100	5.23
October	125,500	4.29
November	138,900	4.29
December	159,900	3.98
January, 1932	90,700	4.33
February	90,500	3.73
March	84,200	3.78
April	84,300	3.24
May	86,000	3.56

Drill-sharpening shop

Argentine Glance Pitch

Details of initial development, in the foothills of the Argentine Andes, of an interesting variety of solidified petroleum, in vein form, the capping and walls of which provide a kerite — an oxidized asphaltite—for use locally as boiler fuel. Subsequent research in the United States led to its classification as a high-quality glance pitch.

A. W. Allen

Map of south-central Chile and Argentina

Camp and garden at Auca Mahuida

IN THE TERRITORY of Neuquen, Argentine Republic, about 120 kilometers from the Contra Almirante Cordero station of the Buenos Aires Great Southern Railway, is an unusual deposit of solidified petroleum, in the form of two veins, one of which, the Escondida, has been the object of development work and technical research. Surface and subsurface surveys indicate an average vein width of three meters over a length of 426.5 meters. The bituminous material, an asphaltite of high purity and uncommon characteristics, has been known for many years in Chile and the Argentine as raphaelite —the name originally given to a substance, also of petroleum origin, discovered at San Rafael, near Mendoza, about 400 miles to the north. The two minerals, however, have little in common; and the name raphaelite is a misnomer when applied to the Neuquen asphaltite, which is a variety of glance pitch, as will be demonstrated later in this article.

An abstract follows from a geological report, by Dr. T. R. Marshall, on the property, which consists of two *minas* and five *estaca minas*, each 900 by 600 meters, on the Escondida vein; and two *minas* and seven *estaca minas*, also 900 by 600 meters, on the Auca Mahuida vein:

Almost the whole area lies in a circular valley surrounded for the greater part by bluffs from 150 to 200 meters high. The south-west section, for 10 kilometers, opens out into the pampas beyond. The valley has been formed by the deep erosion of a plateau. The level of this still persists over enormous areas, and is covered by a sheet of basaltic lava or a layer of coarse gravel known as the Patagonian deposits. The escarpment flanking the north-west side of the valley, which rises to an elevation of 210 meters above the floor of the bay, is capped by several meters of dense black basalt. Further to the north-west, the lava-covered surface gradually rises to the main vent of a volcano, Auca Mahuida, which burst through the plateau of sedimentaries after the strata had been raised by a continental thrust above the gradually receding shallow sea. Extensive erosion has occurred since this period, laying bare the deeper rocks in certain areas to such an extent that a determination is possible of the ages and characteristics of the various strata, down to those of the Jurassic period. The plains extending to the south-east of the Barranco del Palo are only slightly eroded; but in the intervening area, between the base of Sierra Auca Mahuida and the Barranca del Palo, including the area occupied by the property, all but the coarse sandstones of the lowest series of Tertiary strata have been swept away, and the upper beds of the Cretaceous period lie close to the surface. None of the beds exposed in this region is horizontal.

A long-continued continental thrust has folded the deeper strata into geanticlines and geosynclines; and, as the force has continued through Tertiary times, the Tertiary sedimentaries are now tilted to a slight extent—on the property to about five meters per kilometer. In addition to this persistent continental thrust, a minor local thrust from the north has been of considerable severity, but this died away during early Tertiary times, as only the basal Tertiary sandstones have been affected. This minor thrust has produced in the deeper strata a series of anticlinal folds, the crests of which traverse the valley in a direction about north 70 deg. west. Near the anticlines, denudation has exposed in a few places the upper cretaceous strata which dip from 10 to 14 deg. to the south on the south side of the anticline, and also to the north on the other side. These strata are overlain unconformably by the lowest Tertiary sandstones, which have dips of from 2 to 3 deg. north and south. Two veins of asphalt outcrop. The strike of the Auca Mahuida vein is north 68 west and that of La Escondida, north 69 west. The strata dip to the north on the north sides of the veins, and to the south on the south sides of the veins. These veins therefore have evidently been formed in open fissures along the crests of two anticlines. The prolongations of the lines of strike of the two veins are 825 meters apart, showing that the veins are situated on separate and distinct anticlines. Distinct indications of a dome structure are in evidence, as the anticlines pitch to the east or west at different points, and the asphalt apparently outcrops at the tops of two domes. Almost in line with the general strike of the anticlines, 8,150 meters from the center of the Auca Mahuida vein, is a small extinct volcano, which was active at the close of the Tertiary period and was responsible for the formation of open fissures along lines of fracturing, which would naturally be most severe along the crests of the anticlines. The asphalt found in these veins is the solid residue of petroleum, which has been forced up through fissures that have penetrated oil resevoirs below.

Development work, confined to the Escondida vein, was planned to prove continuity, to demonstrate width, and to provide material for examination and

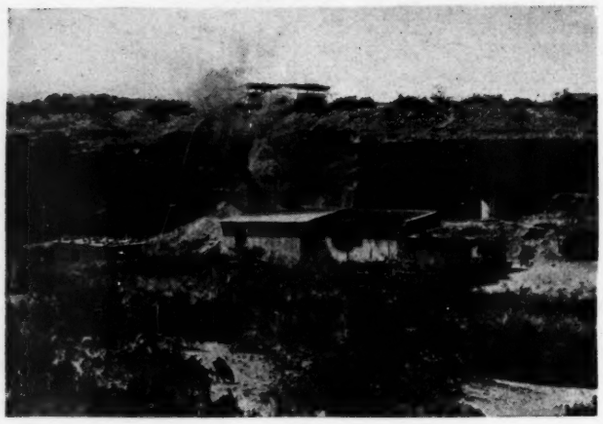


Fig. 2—Water pumping plant, operated by kerite-fired boiler

analysis. Surface showings proved to be of kerite—an altered asphaltite, soft in some places and easily powdered, but elsewhere as hard as and tougher than ordinary coal, particularly when in contact with the unaltered solid petroleum. This kerite proved an excellent fuel for boilers, and is used thus in connection with a steam power plant supplying water for domestic purposes. The extremities of the vein, as disclosed by subsurface exploration, show an ash-like overburden; the kerite in the center region of the vein is clearly in process of oxidation. The assumption was therefore first made that the kerite occurs at or near the surface only, as a result of atmospheric action.

In addition to several test pits, however, a short tunnel was driven, a shaft sunk, and two small stopes were opened by sloping entries in the central portion of the vein. At the lowest depth reached, about 10 meters, examination of the face indicated the presence of small veinlets and larger pockets of the kerite in the asphaltite. These seams or pockets, traced to the wall or roof, were found to act as watercourses. I therefore concluded that the preliminary oxidation of the asphaltite and the formation of the kerite was the result of penetration of water carrying sufficient oxygen to effect the change. The original material is dense and oily, but the various stages of deterioration can be traced: A surface oxidation occurs first, causing a shrinking of the asphaltite and formation of radiating cracks. The water penetrates these, and the zone of oxidation is thereby increased, the same channels being available for the escape of the lighter gaseous fractions of the solid petroleum. Stresses in the material and the degree to which a flow of water can be established determine the rate and degree of penetration of water and oxygen.

By accepting this theory, therefore, one may assume that any narrow vein in contact with water will be oxidized, completely or partly, to the center, which appears to be the case with the Auca Mahuida vein near by. In consequence, it is apparently of secondary economic importance except as a source of fuel.

To establish a tonnage basis on an adequate vein width, development work on the Escondida indicated a zone of probable commercial productiveness in and between two minor hills, to the east and west, respectively, of a center hill. Hillside cuts demonstrated that no indication of the character of the vein proper could be obtained without stripping to a considerable depth. The character of the kerite uncovered in prospecting proved to be a good indicator of the character of the pitch underneath: the harder the kerite, the denser and purer the pitch. Beyond the two outer hills, and in the eastern slope of one of them, a change in vein character is discernible: the kerite is much softer, and the vein wall is burnt and irregular. Seepages of the original petroleum, or the semi-fluid mass, are shown; and a small amount of sandy oil shale, of no commercial significance, is disclosed. The vein matter in some places is mixed with stones; and grass roots were found in the vein at 3 meters' depth in the extension territory. From these indications a differentiation was made between promising and unpromising parts of the vein. Thus was obtained a preliminary vein-width estimate of three meters over a length of 426.5 meters, roughly checking the results of incline core drilling in 1922, which indicated a width of 3.25 meters at 68 meters' depth. An estimate based on such figures indicates positive vein contents of about 520 metric tons of kerite (coal) and 940 metric tons of glance pitch per meter of depth, assuming a volume ratio of one of the former to two of the latter, and allowing for the difference in the density of the kerite as compared with that of the glance pitch. The depth of the vein is likely to be considerable, although positive figures would be impossible to obtain because of difficulty of sample boring. The material is very friable. Rotation of a drill causes complete comminution, inhibiting differentiation between the kerite and the glance pitch where visual examination is not possible. The determination of proved reserves in positive terms would involve the development of an uncommon technique in core drilling.

The pitch was mined by hand—by gad and hammer. The application of a jet of live steam, had it been available, would probably have been cheaper and quicker; but excessive fracturing was avoided because of the concomitant production of fine material that soon oxidizes and so contaminates the sample. Large slabs of the pure pitch, weighing up to 10 kilos apiece, could easily be removed by hand; but such pieces disintegrated into smaller ones overnight, indicating the strains in the material *in situ*. The bulk of the sample shipped came from one working, where the vein width was

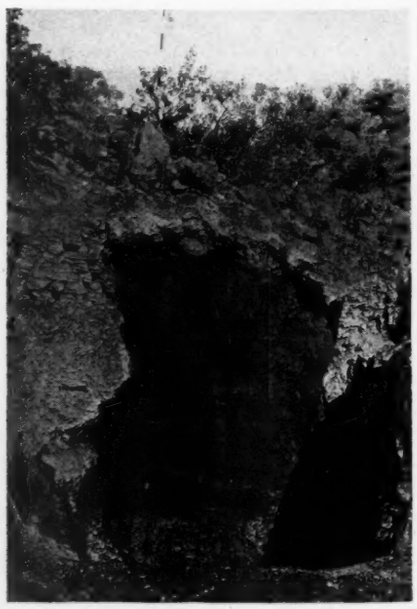


Fig. 3—Exploration pit in hillside, exposing soft kerite above and hard kerite below

Fig. 4—Absence of penetration into the surrounding soil indicates the dense nature of the original fluid bitumen

Fig. 5—Entry to working, with striations of unaltered glance pitch showing in roof and kerite vein at surface

about 3.6 meters, about one-sixth of which on each side was kerite. As this stope was only about six meters below the surface, the natural oxidizing action is obviously slow. The kerite walls stand well, and the pitch roof holds, as a brilliant dome of jet, after the radiating lines of fracture started by the gads have released intervening material.

The illustrations reproduced herewith show that the vein is almost in its original condition, the structure of the top of each end appearing apparently as originally forced upward as a gelled and hardening mass, without reaching the surface. Erosion consequent on exposure is evident in the central portion, however.

From samples of a shipment of about 40 tons of the glance pitch sent to New York an elaborate examination was made, to place the material in its proper category and to indicate opportunities for commercial utilization. Analyses and properties are given in the accompanying table.

The presence of semi-rare metals in petroleum and its natural derivatives gives rise to interesting conjectures. Analysis showed the presence of vanadium and nickel in the unaltered Escondida pitch and in the kerite that formed the walls and roof of the deposit. Moreover, a part at least of the vanadium in the kerite was soluble in water. J. E. Spurr, in discussing a deposit in Peru that contains vanadium sulphide, iron-nickel sulphide, asphaltite, and what has been termed coke, suggests that the original vanadium solutions were hot magmatic solutions, foreign to the asphaltite, and to him they appear to have partly coked it. The presence of vanadium and nickel has been demonstrated in Argentine petroleum, but one must go further back in the assumed evolutionary process to explain the presence there of either or both of these metals in Escondida pitch and kerite. Of interest to note is the fact that, although vanadium is found in and associated with the asphaltite in the Peruvian deposit, it is absent from the coal found in the same district.

Properties of Escondida Vein Material

1.	Color	Black
2.	Fracture	Conchoidal
3.	Luster	Bright
4.	Streak on porcelain	Black
5.	Specific gravity, 20°/4°	1.105
6.	Temperature of initial decomposition, deg. C.	About 315
7.	Melting point, ring and ball, deg. C.	175
8.	Melting point, K. & S., deg. C.	159
9.	Volatility, 6 hr. at 163 deg. C., per cent	0.04
10.	Flash point, deg. C.	Above 338
11.	Penetration at 25 deg. C.	Nil
12.	Carbon, per cent	83.45
13.	Hydrogen, per cent	8.24
14.	Sulphur, per cent	2.27
15.	Nitrogen, per cent	1.1
16.	Oxygen, per cent	4.19
17.	Vanadium and nickel, presence indicated by spectroscope	....
18.	Ash (mineral matter), per cent	0.2
19.	Fixed carbon, per cent	30.3
20.	Iodine value	60.8
21.	Carbenes per cent, under	2.7
	Percentage solubility in	
22.	Carbon disulphide, c.p.	99.81
23.	Carbon tetrachloride, over	97.11
24.	Benzol, c.p.	74.3
25.	Chloroform, U. S. p.	99.76
26.	Toluol, c.p.	84.0
27.	Turpentine	64.0
28.	Petroleum naphtha, 88 deg., per cent	31.6
29.	Gasoline, 86 deg., paraffin base	40.3
30.	Ether, U. S. p.	46.0
31.	Ethyl alcohol (95%)	3.4
	Electrical properties:	
32.	Volume resistivity, ohms, cm	3.1×10^{15}
33.	Effective a-c conductivity, mho. cm	2.1×10^{-12}
34.	Dielectric constant	2.9
	Characteristics of kerite forming capping and walls:	
35.	Ash, per cent	0.2
36.	Volatile matter, per cent	65
37.	Fixed carbon, per cent	35
38.	Specific gravity, 20°/4°	1.22

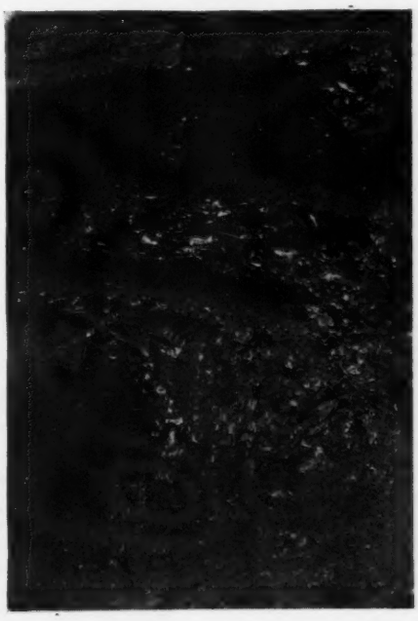


Fig. 7—Glance pitch, and a soft kerite derived from it, a few meters below the surface

Fig. 6—At one extremity of the vein, where no erosion of the deposit has occurred

The catalytic action of certain metallic compounds on petroleum has been established. A mixture of aluminum chloride and petroleum yields, on distillation, light and heavy fractions; the catalyst, removable by washing with water, is found in the latter. A suggestive contribution to the subject has been made by Dr. Murray Stuart, who avers that large quantities of oil may be carried down in association with a mud by reason of the viscosity of the mass and the affinity of the oil for metallic sulphides contained in the mud. Concentration of mineral particles by means of oil may thus prove to be a natural phenomenon of great age, a conjecture all the more plausible when one remembers that flotation oils vary in their affinity for mineral particles of different composition, and that the amount or distribution of vanadium and nickel in petroleums is likewise subject to variation.

An interesting speculation was prompted by a study of the Argentine deposit—to account for the change of the petroleum from the liquid to the solid state (not comparable with surface effects, in the production of a comparatively soft bitumen elsewhere, by loss of the lighter fractions) to form a massive vein of glass-like asphaltite. One factor is noticeable: the mineral contains more sulphur than probably did the petroleum from which it was derived, an addition derived perhaps from gypsum, found in abundance in the district. Is it not possible, however, that the contained nickel and vanadium, under favorable physical conditions, acted as an electrolyte or catalyst, gelling the oil to form a dense though fluid mass, in which condition it was forced up to fill the fissure in which it is now found?

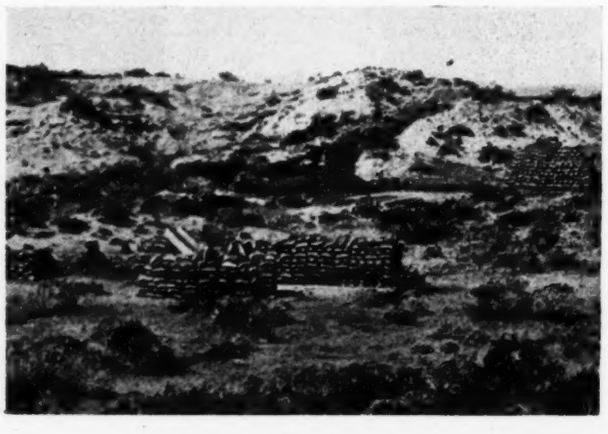
Fig. 8—Glance pitch as mined; the largest piece weighed about five kilos

Subsequent steps in solidification and hardening are understandable as paralleling similar phenomena in colloid chemistry. Exposure of the asphaltite on the surface resulted in oxidation, with the production of a kerite of insignificant solubility in reagents in which the asphaltite can be almost completely dissolved.

The wide variety of bituminous substances found in nature is partly responsible for the difficulty experienced in their proper identification; and much complication in definition has resulted from lack of cooperation among authorities. The literature on the subject has grown by accretion, without discard of theories and classifications unacceptable to the majority of students and investigators. A foundation for theory is the simple statement that a natural bituminous substance is derived from petroleum. Hackford's definition of bitumen is broader that that of the majority, and so is less subject than are many others to adverse criticism: "A naturally occurring organic substance—gaseous, liquid, or solid—consisting of hydrocarbons and the oxy- or thio-derivatives of the same, or of a mixture of all three." Bitumen, characterized by high content of colloidal carbon, is essentially a surface material. When deposited as vein matter it undergoes further change, by heat and pressure, or by a hardening agent such as sulphur, or by the loss of protective colloids, or by catalytic or electrolytic action, or by combination of any or all of such influences, and becomes asphaltite—the name given to a series of solid bituminous substances including gilsonite, glance pitch, manjak, and grahamite. The effect of heat in the formation of these materials is indicated by presence of carbenes, which are insoluble in carbon tetrachloride but soluble in carbon disulphide. As a general rule, the carbene content increases as the classification proceeds from gilsonite (with none) to grahamite (up to 80 per cent). The final step, well illustrated in the Escondida vein material, is the change by oxidation from an asphaltite to a pyrobitumen characterized by high melting point and low solubility. This material, which may be classified as a variety of albertite or impsonite, varies in composition according to whether or not it has been exposed to the atmosphere. Identification of the original vein material is simpler and yet more complicated—simpler because of uniform composition and high purity, and more complicated because of the unconformity of some of its characteristics with known material of a similar type occurring elsewhere. It differs from gilsonite in giving a black streak on porcelain, in a higher specific gravity, and a higher content of fixed carbon. It differs from grahamite in lower specific gravity, lower fusibility, and lower content of fixed carbon. It corresponds in general characteristics to the known properties of glance pitch or manjak, as outlined in Herbert Abraham's "Asphaltites and Allied Substances"; and it doubtless originated from a type of petroleum differing from that which preceded the formation of gilsonite or grahamite. In only one particular does an analysis show apparent divergence from a standard specification for glance pitch—in regard to carbene content. The ordinary tetrachloride test proved difficult to apply, and a modified laboratory technique was developed. A regular increase in the estimate of solubility was noted according to the elaborateness of the test; but, in spite of careful laboratory work, no positive report was possible, and the carbene content was given as under 2.7 per cent.

Identification of the Escondida vein material has been complicated by several factors. Casual examination by various observers in former years apparently failed to differentiate between the Escondida vein and the Auca Mahuida vein. A mixed sample from both would be misleading and valueless. Earlier observers also apparently failed to realize that the capping over the Escondida vein was an altered asphaltite, not representative, in any respect, of the original and unaltered vein underneath. Furthermore, laboratory tests indicate a high absorption of oxygen by finely ground Escondida pitch, indicating that the material is sensitive to atmospheric conditions, and that a true analysis is possible only if undertaken on lump samples. Disregard of these facts was doubtless responsible for designating the Escondida material as raphaelite — the name given to another bituminous substance, probably a type of grahamite, found elsewhere in the Argentine, and mentioned at the beginning of this article. A published analysis of raphaelite, by Dr. J. J. Kyle (*Chemical News*, Vol. 46, 1892), indicates a specific gravity of 1.173, an ash content of 0.63 per cent, and a fixed carbon content of 47.81 per cent. The sample from which the analysis was made may have been partly oxidized, or it may have contained some naturally altered asphaltite; but the reported content shows how radical is the difference between raphaelite, as found in the Province of Mendoza, and any combination of the products comprising the Escondida vein. To emphasize this distinction, I have suggested that the Escondida material be named neuquenite, after the territory in which it is found.

Confirmation of the high quality of neuquenite, and the correctness of classifying it in a class between and distinct from the lower-valued gilsonite and the still lower-valued grahamite, is seen in the fact that the shipment of 40 tons from which samples were taken for research and identification, together with 20 tons received via Europe—representing most of the output during development as outlined—was disposed of in New York for 3½ to 4c. per pound



Central, exposed portion of vein, leading to survey flag at upper-right corner

Long Slope Mining at Anyox

Frederick S. McNicholas

MINING at the Hidden Creek mine of the Granby Consolidated Mining Smelting & Power Company at Anyox, B. C., has been done by glory-hole, shrinkage, spiral stope, and "long slope" methods. The glory-hole and shrinkage methods follow standard practice. The spiral-stope method has been associated with Granby mining practice for more than thirty years and has been frequently described (see Peele, p. 564, and *Transactions* C.I.M.M., Vol. XXXII, p. 187). For purposes of comparison the following condensed general description of it is given:

The usual development by suitable drifts is made. Stope blocks with intervening pillars (40 to 60 ft. thick) are laid out. Manway raises are driven in the pillars from each level to the level above. Chute raises on 50-ft. centers are driven in the stope block from the drifts a distance of about 15 ft. From the top of these chute raises, spiral raises or, usually, spiral stope (10 ft. high by 25 to 30 ft. wide) are driven to the next level above, steep enough for the broken material to run by gravity down the inside of the spiral and flat enough on the outside of the spiral to maintain a trail for entrance to the face. The finished spiral resembles a huge corkscrew with a manway trail maintained under cover at the periphery. At the extremities of the swings of the spiral, connections are made to the pillar manway raises, at each end of the stope. Subsequent work consists in widening the spiral, benching, and breaking down bottoms and braces.

The principal objections to spiral stoping are:

1. A large proportion of ore broken is in high-cost "cutting out" stopes, with a resulting high over-all mining cost.

2. The difficulty of making the spiral swings within the ore limits, inasmuch as the limit is one of assay only and low-grade patches may occur in the main orebody.

3. The difficulty of coordinating current work with previous work that has defined ore limits. Since the mining is not done in a systematic manner and is progressing in a constantly changing direction, irregular braces are left, the location and size of which are difficult to determine accurately from existing engineering data when they become inaccessible.

4. Removal of irregular braces is often dangerous, difficult, and expensive.

5. Possibility of leaving some ore unmined (irregular legs and braces may not be recovered should caving occur).

6. The method does not lend itself to routine operation, and is greatly dependent upon the personal element.

A variation of the spiral-stope method, called "long slope," has been adopted wherever practicable, to obviate some of the objectionable features of the spiral method. This method (Fig. 1), as used, requires the usual development as previously described, and consist essentially of driving parallel inclined stopes (+ 38 deg.) through the orebody, parallel to the strike of the ore in narrow ore widths and across the strike in wide ore widths, leaving a parallel inclined brace

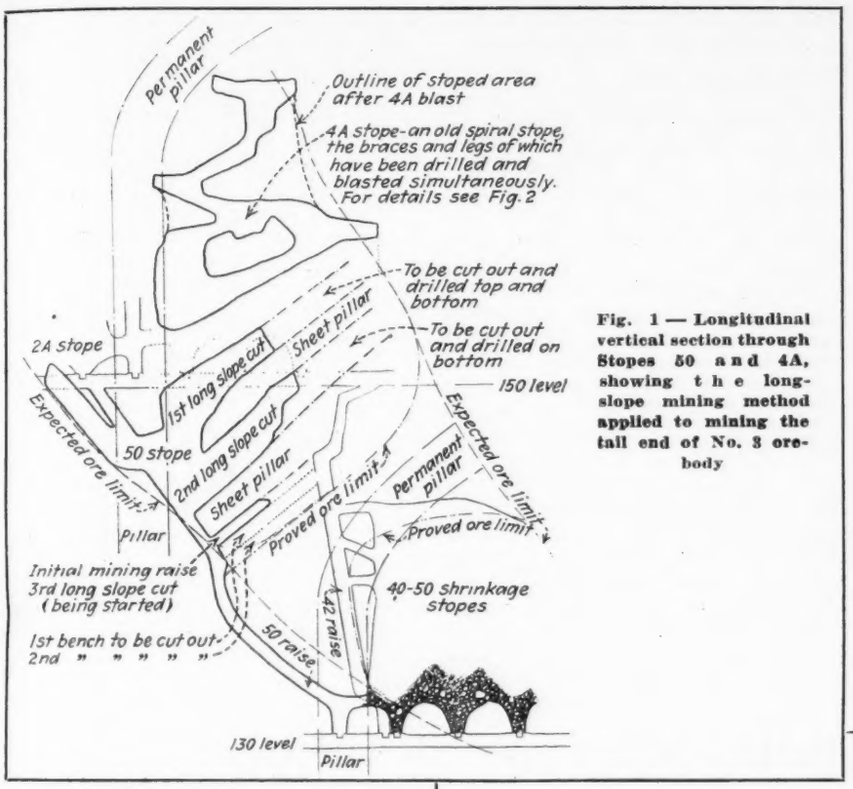

Permanent pillar
Outline of stoped area after 4A blast
4A stope-an old spiral stope, the braces and legs of which have been drilled and blasted simultaneously. For details see Fig. 2
To be cut out and drilled top and bottom
To be cut out and drilled on bottom
2A stope
Sheet pillar
1st long slope cut
150 level
Expected ore limit
50 stope
2nd long slope cut
Sheet pillar
Permanent pillar
Expected ore limit
Proved ore limit
Proved ore limit
Pillar
Initial mining raise
3rd long slope cut
(being started)
40-50 shrinkage stopes
1st bench to be cut out
2nd " " " " "
42 raise
50 raise
130 level
Pillar

Fig. 1 — Longitudinal vertical section through Stopes 50 and 4A, showing the long-slope mining method applied to mining the tail end of No. 3 orebody

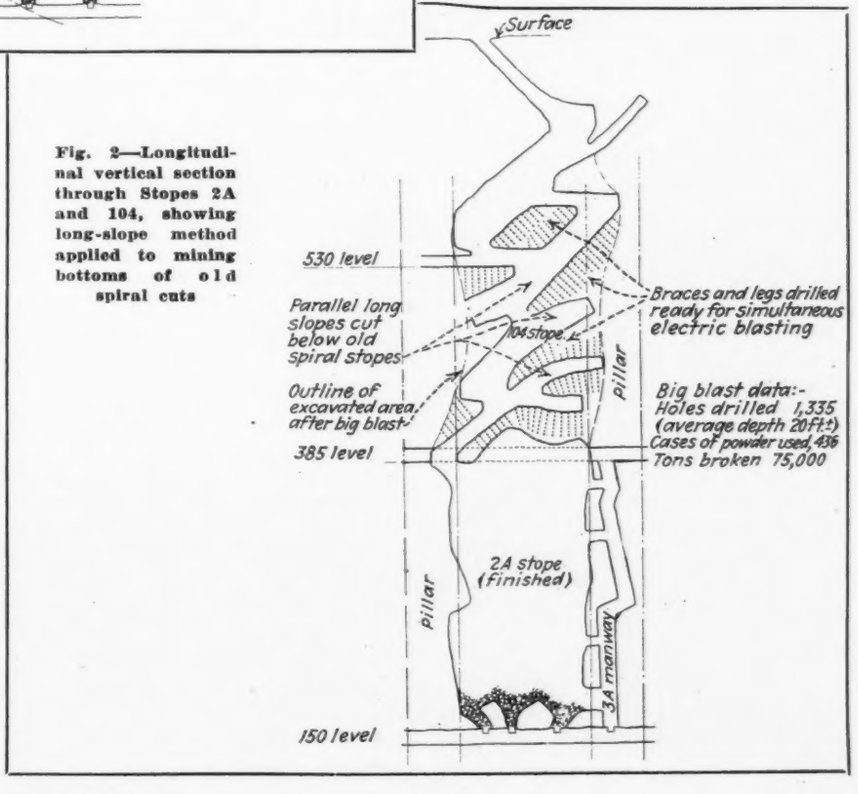
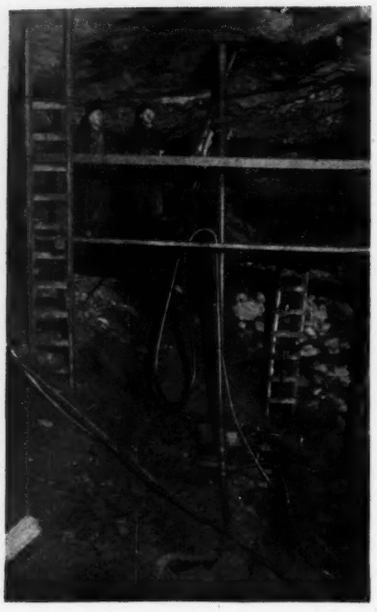


Fig. 2—Longitudinal vertical section through Stopes 2A and 104, showing long-slope method applied to mining bottoms of old spiral cuts



Comparative Data (Approximate), Spiral Stope and Long Slope Mining, Hidden Creek Mine, Anyox, B. C.

	Cutting Out Stopes		Benching		Removal of Pillars		Total	
Mining Method	Per Cent of Total Tonnage	Tons per Machine Shift	Per Cent of Total Tonnage	Tons per Machine Shift	Per Cent of Total Tonnage	Tons per Machine Shift	Per Cent of Total Tonnage	Tons per Machine Shift
Spiral Stope....	25	20	50	80	25	125	100	76
Long Slope.....	15	25	45	80	40	125	100	90

The rates of 76 and 90 tons per machine shift show an 18 per cent increase in tonnage per miner machine shift in "long slope" mining compared with spiral stoping.
Direct mining costs are approximately in direct proportion to tonnage per machine shift.

over the back of each stope of such thickness that it may be mined as one bench with 20-ft. holes.

Starting at the top of the desired ore section, the initial stope (10 ft. high and up to 80 ft. wide) is mined to the established limits, and the back is slabbed, arched, and trimmed to give maximum strength and safety. The bottom is then benched, with drifter drills, tripod-mounted, in a series of benches, until judgment indicates that further benching is dangerous. As benching is the cheapest method of breaking ground at Anyox, an effort is made to secure the maximum amount of such work. In some ground, benching may be carried with safety 70 or 80 ft. from the back.

After benching is completed, a 30-ft. bottom is left under this stope and another parallel stope is driven and benched in like manner.

The process is repeated until the entire block of ground is prepared, leaving a series of parallel inclined braces. These braces are then mined, starting at the bottom of the prepared section, by drilling, from tripods, 20-ft. holes downward from the top of the lowest inclined brace and blasting in sections electrically, or if the brace is weak the entire brace is drilled and blasted in one simultaneous blast electrically.

The method is flexible, but as the height and width of each stop depend upon the character of the ground, the exact interval between stopes cannot be anticipated definitely. At all times an effort is made to carry benching as far as possible with safety. In other words, the amount of benching that may be done in the upper stope must be determined before the next lower one is started. A vertical leeway of about 10 ft. in the location of the succeeding understope results, inasmuch as one cut (10 ft.) may be taken off the back, or not, as conditions require.

In undercutting an old bottom that cannot safely be drilled from above, or if benching in any stope is carried so far that subsequent drilling from the top of the inclined brace left under this stope is dangerous, the undercutting stope may be benched only once, leaving a stope 20 ft. high between two inclined braces from which to drill 20-ft. holes in bottom and back (from 18-ft. bars) for the removal of the braces in sections simultaneously. Where the stope cannot be carried the desired width, a rib pillar may be left between two cuts of the same stope which can be taken with the braces. The method has been used for mining old spiral stope bottoms (Fig. 2), in mining the tail ends of orebodies (Fig. 1), and in conjunction with existing work.

Approximate comparative performances with the two methods are shown in the accompanying table. The long-slope method is particularly suited to mining the lower ends of orebodies where the lower ore limits are not definitely known because the work is carried downward from known ores.

■

Non-Metallic Mineral Flotation

Dipl. Ing. Oskar Sommer

FLOTATION PROCESSES for oxidized minerals are influenced by two fundamentally different lines of thought—sulphidizing of the mineral by the adsorption of a sulphur compound, and direct flotation by addition of higher fatty acids or salts. Some of the methods in the latter group are comparable with the treatment of sulphide ores by addition of xanthates, which is based on an increase in already existing differences in wettability by adsorbed compounds of low solubility. Similar phenomena may be assumed to take place in the flotation of oxidized calcium compounds, which is made possible by adsorption of heavily soluble compounds of fatty acids. In both processes the minute quantities of collector required—from 50 to 100 grams per metric ton—indicate that related forces are brought into play.

Contrast to this behavior is, however, offered by oxidized minerals such as cassiterite, kaolin, and others. My studies with flotation methods for kaolin make reactions of this type appear questionable. The methods referred to involve the agitation of the mineral with water glass, followed by addition of an acid salt. A fatty salt is then added as frother. Addition of water glass and acid salt serves to depress the gangue and thus increase the difference in wettability. Gerth has explained the phenomena involved in the treatment of a cassiterite-quartz mixture in the following way: "As reason for the depression of the quartz must be considered a wettable layer of silica gel on the quartz particles, rendering the quartz particles amorphous and thus non-floatable." (*Arch. Erz. Met.*, 2, 1932.) Formation of the gel on the quartz surface has been proved by increase in weight, coloration, and microscopic examination. To suppose that the related chemical properties of quartz and silica gel should result in such an attachment may be considered reasonable.

With this condition in mind, experiments were begun with a raw kaolin that was contaminated with quartz and small amounts of mica. As all of these compounds are silicates, the question arose as to whether or not the formation of gel would take place exclusively on the quartz particles. Experience had demonstrated that best results were obtainable with a neutral pulp (pH 7). Addition of equivalent amounts of water glass and aluminum chloride, the acid salt used in this case, therefore seemed correct. Assuming for water glass the composition Na_2SiO_3, the stoichiometric equation is: $Na_2SiO_3 : AlCl_3 = 1.37:1$. Tests were therefore made with the following additions: Sodium water glass, (1) 1.37 and (2) 6.85 kg. per ton; aluminum chloride, (1) 1.0 and (2) 5.0; and soap, 10.0 kg. per ton in both instances.

Addition of reagent was made in the sequence given, with 5 min. stirring after each addition. Both tests gave negative results, showing that silica gel was formed on the surface of all three

Translation, by B. H. Strom, of a report from the Laboratory of Ore Concentration of the School of Mines, Freiberg, Germany, in *Metall u. Erz*, No. 13, July, 1932.

minerals. This, as already suggested, appears reasonable in view of the fact that all three minerals are silicates, related to the silica gel.

Previous studies of the colloid chemistry of kaolin suspensions lead to the assumption of a pure colloid-chemical reaction between the minerals of the raw kaolin, dispersed by the water glass, and the coagulating action of a salt with a polyvalent cation, thus confirming the rule in flotation practice that everything that can be coagulated may also be floated. In a second test the amount of aluminum chloride was therefore increased to 9 kg. per ton, whereby an extra large amount of heavily mineralized froth was formed, proving the correctness of the reasoning.

This result throws new light on the process. The tests indicate that coagulation plays an important part. Whether the resulting floating action may be ascribed solely to the enlargement of the particles through flocculation, or to the accentuation of the small differences in the wettability of the individual oxidized minerals by replacement of the water layer with a coagulating agent, will not be discussed here. Conditions may be controlled by varying the degree of coagulation. The dispersion already referred to thus serves only to break up the flocculated aggregates into their primary constituents, to prevent contamination of the concentrate. The type of dispersing agent used is therefore of no importance, whether it be an electrolyte, as sodium hydroxide or ammonia, a protective colloid, a combination of the two, as water glass, or even fatty salts and soap. Neither seems to be the type of coagulating agent that influences the result. That certain coagulating agents have a decided selective action upon some minerals must not be disregarded. Experiments with selective precipitation should find a fruitful field. Such action may possibly be the underlying reason why kaolin floats, although it represents the finest particles in the mineral mixture. That the largest particles coagulate first in such systems is a well-known fact.

With this explanation the question is left open: Can the method be classed with the process for treatment of sulphide ores?

The effect of the coagulating agent—utilizing the existing differences in wettability—is undoubtedly different from that of the xanthates. A further step toward the perfection of the flotation process will have been taken when chemical research has succeeded in finding compounds for oxidized minerals, which, by adsorption of the mineral to be floated, will reduce its wettability to a fraction of its former value, thus making it more floatable than the associated minerals. Until that time, however, the method described is the best one available.

Seen from this point of view certain phenomena observed by Gerth are confirmed. Best results are clearly obtainable with a particle size below 0.088 mm.; with larger sizes the force of gravitation greatly exceeds the effects derived from adsorption. Why flotation of Altenberger ore first resulted in a tough red-colored froth is also explained; the small particles possessing the greater force of adsorption attracted all of the coagulating agent. An equally good explanation is that saturation with dissolved salts with polyvalent cations caused flotation as soon as the collector was added. Presence of such coagulating agents will also serve to explain why this slime could not be dispersed with sodium carbonate, as the dispersing property of this salt is low.

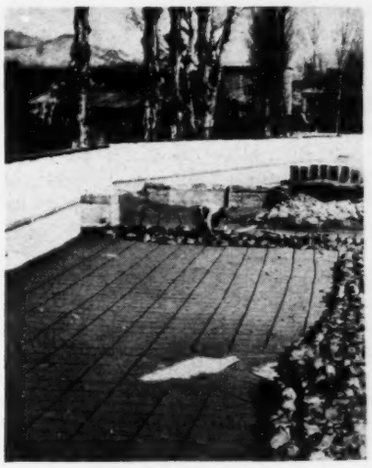
■

Processing Ocher in France

OCHER is a limonite or hydrated sesquioxide of iron and may be mixed with clay, silica, or other impurity. It is colored by the iron oxide and furnishes to painters and artists a gamut of tints from yellow to brownish red. The most celebrated deposits are found in France in the region of Apt, in the department of Vaucluse, where they are actively exploited either in open cut or in underground galleries. In the former operation the ocher is excavated in benches 13 ft. high. Underground, it is taken out in 10-ft. rooms, separated by 20-ft. pillars.

From the mine, the ocher is carried in small carts to the washing plant. The ocher content of a cubic meter of this earth is only about 440 lb., so washing is necessary. About 1.5 gal. of water is required to wash 1 lb. of raw material and rewash the residue. The ocherous earth is mixed with water and the mixture run, with the ocher more or less in suspension, into decanting basins. Agitation is then employed to facilitate the further separation of the ocher and the impurities and the material is allowed to flow through a sloping trough or gutter into a settling pond, where the sand settles out and the fine suspended ocher overflows with the water into the other settling basins.

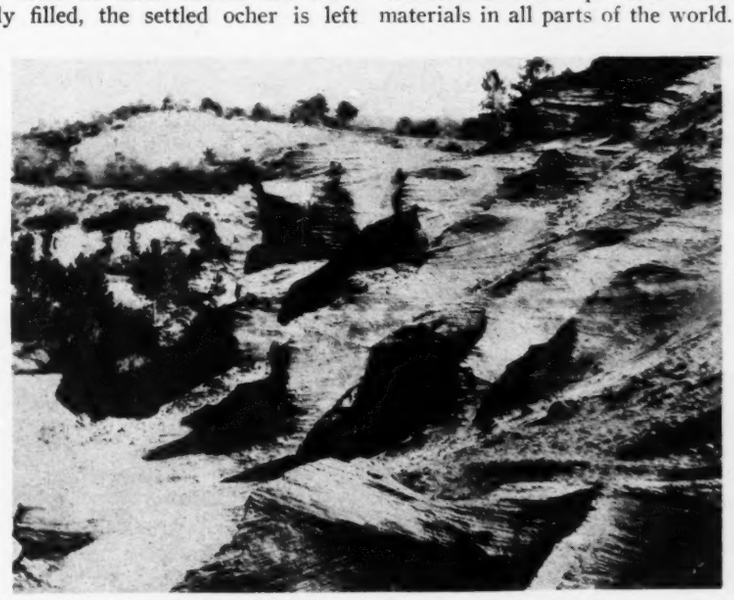
As soon as these basins are sufficiently filled, the settled ocher is left



Cutting and drying the ocher

to dry. When dried it is cut into blocks, as shown in an accompanying photographic reproduction. The blocks are then finely ground and the ground material is softened and packed in barrels, compression being used because the ocher is very light.

This industry employs a large number of men. Every month in the year the quays of Apt are encumbered with barrels of ocher, awaiting shipment to wholesale dealers in painters' and artists' materials in all parts of the world.



Ocherous clay at Rousillon, near Apt, France

Improved Mine Fire-Fighting Equipment

Henry Briggs

AMONG respiratory apparatus of the hose type, used in rescue work underground, is the ordinary smoke helmet (Fig. 1). It is a modification of the diver's dress. The head is incased in a leather helmet provided with windows, and the air is fed to the helmet through a wide-bore hose from a bellows, usually of the double-acting type. This apparatus is a familiar item of equipment at many mines and at rescue stations in all mining districts. Though limited in scope, it has proved of service in fighting underground fires. Recent improvements in design, with which this article deals, have for their object the mitigating of certain inherent disadvantages as much as possible, so as to improve the convenience, security, and range of the apparatus.

On occasion work has to be carried out in foul air when the more elaborate self-contained rescue apparatus is unobtainable except after dangerous delay, or is unnecessary because of the shortness of the distance of the locus of operations from fresh air. Chemicals are not necessary. The appliance is simple. Prolonged training in its use is not essential. These are a few of its favorable features.

An obvious disadvantage in the old smoke helmet is that, by inclosing the head, the wearer's vision is restricted and his hearing impaired by the hissing of the air entering the helmet. As the illustrations (Figs. 2, 3, and 4) show, this difficulty is overcome by substituting a mouthpiece and noseclip for the helmet. The adoption of a mouthpiece led to the invention of the "equalizer"—the first improvement on the smoke helmet for mining purposes—an apparatus in which the bellows are dispensed with, the user drawing his own air supply through the pipe. They are provided with an attachment for converting the pulsating flow generated by the lungs into a uniform flow in the pipe. The original equalizer (called the Antipoys) was designed by J. S. Haldane in 1914[1], the equalizing attachment being a wide-bore corrugated rubber tube (see Fig. 2) introduced between the mouthpiece and the hose. The corrugated tube collapses, like a concertina, during inspiration; during expiration it elongates again and so fills itself with air. In 1918 I introduced a form of equalizer (Fig. 3) in which a bellows-shaped bag, connected to the hose, is carried on the back. The bag is constructed of balloon fabric mounted on strips of steel, which tend to keep the bag distended. When the wearer exhales (discharging the air through a non-return valve into the atmosphere), the bag, partly flattened during the last inspiration, recovers itself, and in so doing draws air through the hose. On the next breath, air is sucked in both from the bag and from the pipe. The equalization is effective, the rate of flow in the pipe being almost constant.

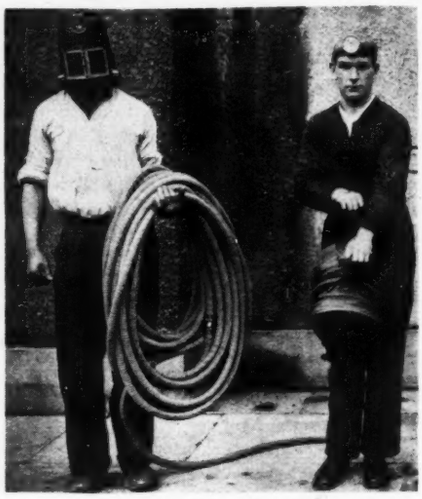


Fig. 1—Smoke helmet, bellows, and large-bore hose

Unless some means of regularizing the flow is adopted, the lungs cannot supply themselves through a long tube with the air needed when physical work is being done; they are unable to overcome the peak resistance caused by intermittent flow. For example, suppose 50 liters of air per minute to be required, the ¾-in. hose being 90 ft. long. Without equalization, the lungs would be called upon to exercise a peak suction equivalent to 8 in. of water column—a suction impossible of realization; yet a uniform flow of this volume can be maintained under a water gage of less than 2½ in., and this difference of pressure the lungs have no difficulty in maintaining.

Fig. 2—Haldane's equalizer with large-bore hose

Both the Haldane and Briggs equalizers, officially approved for British mines, are used in connection with hose up to 120 ft. long. They are, however, negative-pressure appliances, depending for security on sound materials and on regular testing. Even then, a negative-pressure device may become dangerous if the tube is cut, the rescue man being unaware that anything was wrong. The equalizer, as does the original smoke helmet, requires a heavy, large-bore hose. The limit of range of both appliances is determined by the weight of hose a man can manipulate. In mines that limit is about 120 ft.

The latest design (Figs. 4 and 5) greatly extends the range by adopting a thick-walled rubber tube of only $\frac{5}{16}$-in. bore, the air being pumped through it at a gage pressure (usually) of 3 to 7 lb. per square inch. The idea behind the apparatus is the same as that controlling high-voltage electrical transmission—the higher the pressure, the smaller is the conductor necessary to convey a given current. As the strength of a pipe is inversely as its diameter, and as the apparatus is emphatically under positive pressure, a wider serviceableness is combined with greater security.

The air flows into a 6-liter bag carried on the back, and is taken thence through a mica inhalation valve into the lungs. Expiration is direct to the atmosphere through a rubber spear valve, which also effects the discharge of saliva. Excess of air is always supplied, any that is not wanted blowing off through the spear valve without incommoding the operator. Of interest to readers of *Engineering and Mining Journal* is the fact that the notion of applying a small tube and air under pressure was taken from its pages: it was suggested by a description, published over 24 years ago, of fire fighting at the Homestake mine, in South Dakota, where the men operating the water nozzles were provided with home-made leather helmets fed through small-bore hose coupled to the compressed-air line[2].

[1]This, and other apparatus described in the article, is made by Siebe, Gorman & Company, Ltd., London.

[2]*Eng. & Min. Jour.*, March 28, 1908.

Fig. 3—Briggs equalizer with large-bore hose

The pump (Fig. 4), made of duralumin, weighs the same as 120 ft. of ordinary smoke-helmet hose; but, whereas the pump is stationary at the fresh-air base, the hose has to be carried by the rescue man. The pump is double acting; it has a dust trap and annular receiver and is fitted with a folding handle for ease of transport (Fig. 5); and it is worked by a back-and-forth rowing movement while the operator sits on the cylinder. The tube has no wire reinforcement; it is non-springy, is quickly coiled up and unwound, weighs 16 lb. per 100 ft., and is effective for a range of at least 250 ft. It is provided in three lengths, of 100, 75, and 50 ft., and these lengths may be coupled up one, two, or three at a time, as required. The couplings have jam screws, to prevent accidental loosening.

A pressure gage is attached to the pump receiver. The pump man keeps the gage continually under observation: it warns him at once of anything untoward. A sudden rise of pressure indicates a check; and a sudden fall, a leakage of air supply.

When the apparatus was first described, a ¼-in. tube was in use[3]; later, a bore of $\frac{5}{16}$ in. was found preferable, the slight increase in size insuring a considerable reduction in resistance and thereby easing the work of the pumper. The accompanying table, relating to the latter tube, gives the volumetric supply in liters a minute for different pressures and lengths of tube.

To maintain a pressure of 5 lb. is so light a task that the pump man is able to continue his work for an hour without pause. Half-hour spells at the pump are suitable when the pressure is 7 lb., and at 10 lb. two men should be put on the pump. A sufficiency of labor is always available at a fresh-air base when such apparatus is being used underground. Volume of air required depends on the nature of work. The

Delivery Through 5/16-In. Hose

Gage Pressure, Lb. per Square Inch	Air Supply, Liters per Minute— 100 Ft.	150 Ft.	200 Ft.	250 Ft.
3	32	26	..	..
4	40	32	26	..
5	47	39	32	26
6	54	45	37	31
7	60	51	42	35
8	66	57	47	40
9	72	62	52	44
10	..	67	57	49

rescue man is given a small hooter, so that he may signal for more air should the bag flatten or should he commence more arduous labor. One of the hardest tasks a man using apparatus of the hose-pipe type is likely to be called upon to perform is that of shoveling rock and throwing it up and over a wall 4 or 5 ft. high. A fit man, adept with the shovel, generally needs about 27 liters (1 cu.ft.) of air a minute for that job. A heavy man, an unfit man, or a man unaccustomed to shoveling, may demand one-third more. Thus it is necessary to be prepared to pump about 36 liters a minute. The table indicates that this supply (for most purposes an over-generous one) is obtainable at a pump pressure of 5 lb. when 150 ft. of tube are connected, and that the corresponding pressures for lengths of 200 and 250 ft. are 6 and 7 lb., respectively. For lighter work, such as building a brick stopping, the pump is capable of feeding two men, connected in parallel, each with 100 ft. of hose.

Certain advantages have already been claimed over the old smoke helmet, notably a 100 per cent increase of range, abolition of the helmet, and greater security attending the use of a pressure gage as indicator of a cut or a choke in the pipe. The greater length of tube now available enhances the safety of operation: the pump can be put farther away from the foul air. Experience at underground fires—where the fumes may surge backward and forward for dozens of yards along a road, and where dangerous secondary ventilation circuits are likely to be created—indicates that any appliance of this kind operating with 120 ft. or less of hose does not provide adequately for the security of the man at the inbye end.

Fig. 4—High-pressure apparatus, with small-bore rubber tube

Fig. 5 — High-pressure apparatus with pump packed for transport

Many tests have been carried out in Edinburgh to compare the serviceability of the ordinary ¾-in. wired hose with the narrow-bore tubing used with the new apparatus. In some of these a loaded mine tram was run over the pipes. The result was always the same: the smaller tube was uninjured, but the larger one was permanently flattened and usually cut. In other trials, rock walls were pushed over on the two tubes. Again the bigger tube nearly always suffered serious injury and the narrow one nearly always escaped. How insecure is the ordinary hose was revealed during a practice: A man accidentally dropped a sledge hammer from a height of about 8 ft.; the head struck the hose; the wire was cut and the jagged ends pierced the rubber, and the tube was flattened and slit open.

The heaviness and unhandiness of the ordinary hose have been accepted as drawbacks for many years. Mere weight, by limiting the range, involves danger of the fresh-air end being in-

[3] *Trans. Inst. Min. Engs.*, Vol. 82, 1932, p. 386.

vaded by foul air; and as that hose is now found to be subject to other risks, the question of its continued use for breathing apparatus in mines calls, I suggest, for critical examination.

One may argue that the new apparatus suffers equally with the old from the drawback that the user is likely to be caught if his tube is buried beneath a fall. But the small tube is not easily cut by falling rock, although it may be squeezed flat. The risk that a fall will cause stoppage of the air supply is reduced by insisting that the man lay his tube as near the side of the road as possible—a precaution all the more necessary if tramming is proceeding. Incidentally, the best place to put the tube for a right-handed man is close against the right-hand side going in. On returning, it will then lie most conveniently for looping over his left arm (Fig. 4).

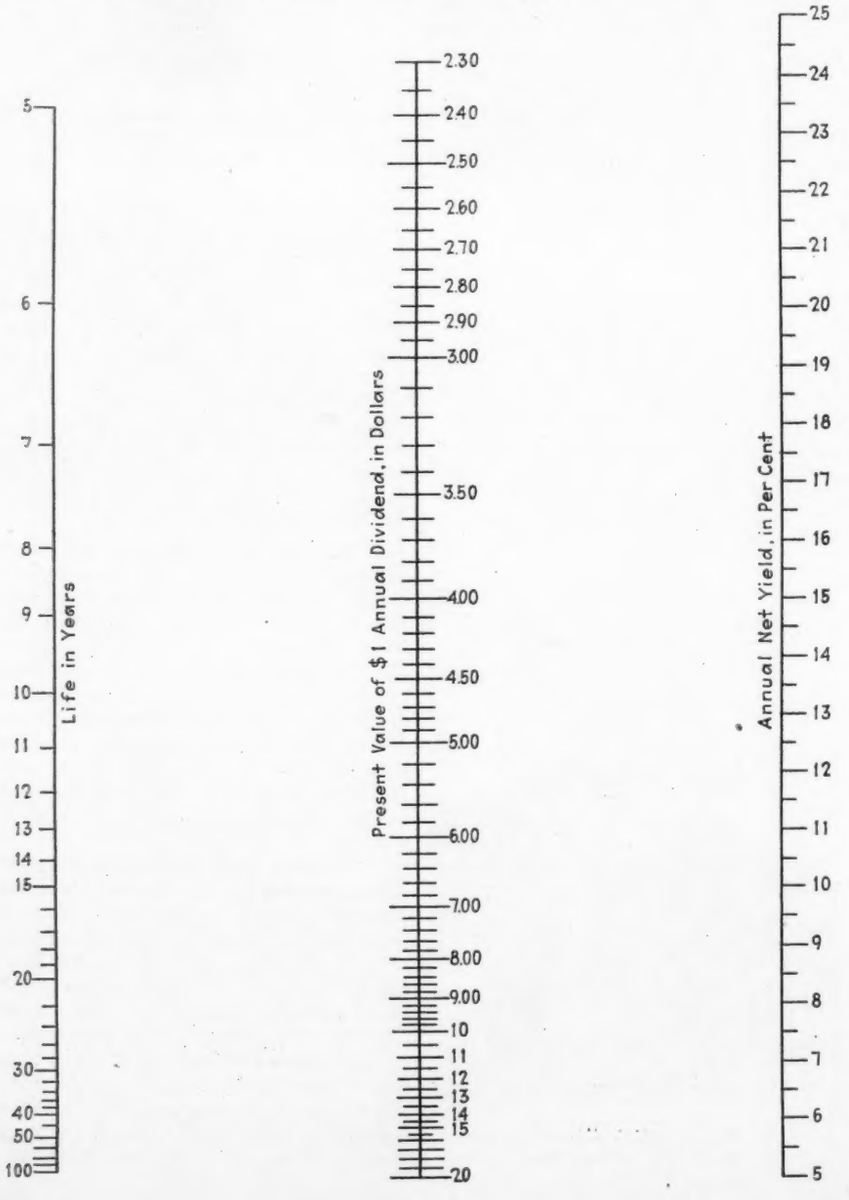
But even if the tube remain open to the passage of air, the man may be unable to release it from under a fall of rock. Acting on a suggestion of Dr. J. S. Haldane, we are now endeavoring to reduce the hazard. Dr. Haldane pointed out that, at a mine fire, foul air may be breathed for a short time without serious effect. "Of course," he said, "it is unfit for a man to breathe for a long time, but if he came out at once it would be perfectly safe. Some people are terrified at the idea of breathing the air of such a place even for two or three breaths, but you can take several breaths of the worst air, where there is no oxygen, without danger." The problem here envisaged thus turns out to be twofold: the designer of a hosepipe apparatus has to provide a harness that may quickly be thrown off, and the trainer has to instruct the men that, in an emergency, the user should discard the apparatus, and, breathing the air of the place, promptly make his way back to fresh air.

[4] *Ibid.* p. 392.

■

Amortization Chart

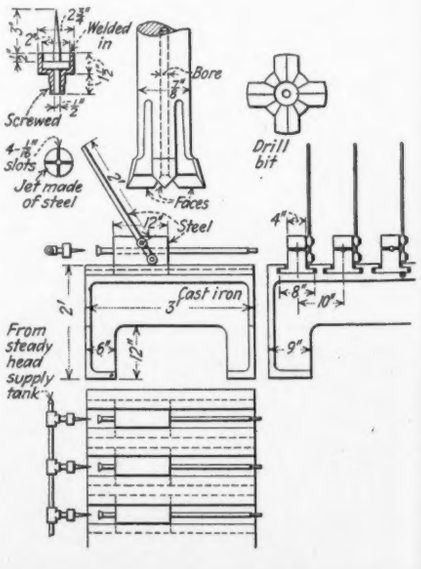
THE accompanying chart, prepared by Nathaniel Herz, chief chemist of the Homestake Mining Company, Lead, S. D., provides easy means to apply the principles and formulas in respect to redemption of capital and interest, as outlined by Herbert C. Hoover in "Principles of Mining" (McGraw-Hill Book Company). As an example: Assuming sinking fund at 4 per cent, life, ten years, and desired yield, 8 per cent, the present value of dividends at $1 per annum would amount to approximately $7.85.



Developments in Rock-Drill Tempering

ORDINARY METHODS of tempering drill steel by quenching permit water to come in contact with the bore of the bit, resulting in a tendency to cause fractures at the section where strength of the steel should be at a maximum. A tempering machine to insure quenching without contact of water with the bore is in use on the Witwatersrand. This machine, shown in the accompanying sketch, permits as many as six drills to be tempered simultaneously, and consists of a cast-iron grooved base in which drill clamps slide up to water jets. A thin, tapered steel pin closes the drill bore as water impinges on the bit faces from four slots in the jet. By moving the lever back, the top half of the clamp is lifted and a drill inserted; on moving the lever forward the clamp grips the drill, and the complete set moves up to the jet.

Many drills can be dealt with at once. All water used is collected and returned to a steady-head supply tank, insuring even water pressure. The machine is of simple manufacture.



Cost of Mining by Caving

Morris J. Elsing

ACCORDING to the U. S. Bureau of Mines, the year 1929 saw 350 underground metal mines in the United States, each of which produced annually more than $100,000. Production of these mines totaled 98,480,455 tons of ore. Of this amount seven mines, using the block-caving method, produced 18,576,196 tons, and 17 mines, using the sublevel-caving method, produced 7,303,976 tons. The open-stope method, with an output of 36 per cent of the total, led in production. The caving method was second, with a production of 26.5 per cent, of which 19 per cent was by block caving and 7.5 per cent by sublevel caving. In addition to ranking second from the point of view of production, block caving was lowest in cost of the six primary underground methods. Only one mine, Alaska Juneau, using a modified shrinkage method, has a lower cost per ton than is shown by the average for block caving.

The Miami mine of the Miami Copper Company, at Miami, Ariz., has the distinction of having both the lowest cost and the greatest annual production of all of the block-caving mines. Production started early in 1911 and has been continuous since then. The first method employed was shrinkage with caving of pillars. Shrinkage stopes and pillars were 50 ft. wide, 200 to 500 ft. long, and 125 ft. high. Before the drawing of ore began, pillars were undercut on sublevels and caved and drawn with the shrinkage ore. This method produced 2,230,577 tons. For mining the highest-grade ore, the top-slicing method was used, and a total of 4,524, 347 tons mined by this method. In 1916, each of these methods produced about 3,000 tons per day. By the end of 1919, top slicing was stopped and a new undercut block-caving method produced all the ore. With present practice, ore is recovered by undercutting blocks 150x150 ft. and 300 ft. high.

Ore consists of secondarily enriched cupriferous pyrite which occurs as seams and veinlets of chalcocite and also disseminated through altered brecciated silicified schist. The ore varies from a hard silicified to a soft kaolinized schist. The orebody is 3,500 ft. long and 2,700 ft. wide, with an average thickness of 325 ft. Barren capping varies from 250 to 500 ft. thick.

Table I—Block-Caving Costs Per Ton

	Miami, Ariz. Annual Report	Inspiration, Ariz. (Partial Costs). I. C. 6169	Ray, Ariz. I. C. 6167	Morenci, Ariz. I. C. 6107	Ruth, Nev. E. & M. J. 11/7/25
Stoping					
Labor	.....	$0.146	$0.165	$0.105	.....
Supervision	.....	0.007	.....	0.040	.....
Explosives	.....	0.017	0.044	0.034	.....
Timber	.....	0.031	0.021	.....	.....
Air drills	.....	0.005	.....	0.045	.....
Power	.....	.....	0.005	.....	.....
Miscellaneous	.....	0.002	0.008	.....	.....
Total stoping	$0.136	$0.208	$0.243	$0.224	$0.340
Development	0.100	0.211	0.104	0.132	0.300
Transportation	0.086	0.144	0.111	.....	0.138
General	0.046	0.012	0.057	.....	0.393
Miscellaneous	0.031	0.008	0.162	.....	0.148
Total mining	$0.399	$0.583	$0.677	.....	$1.319
Tonnage	6,124,993	4,879,646	3,243,159	1,548,258	850,000
Year	1930	1928	1928	1928	1924

Costs per ton shown in Table I are taken from F. W. Maclennan's splendid description of the method published as Technical Publication No. 314 of the American Institute of Mining and Metallurgical Engineers. Details of the cost of mining for the period from 1926-1930 are shown in Table II.

Inspiration Consolidated Copper, Miami, Ariz.—Production at Inspiration started July 1, 1915, and has been continuous since then, except when the mine was shut down during 1921-22. Enriched chalcocite ore occurs in a crushed altered silicified schist and granite. The zone of enrichment is from 50 to 575 ft. thick, with an average of 200 ft. The orebody is 8,000 ft. long and averages 800 ft. wide. Depth of overburden is 100 to 500 ft. The bottom of the orebody is reasonably regular and is either flat or slightly inclined. The orebody is well fractured, which makes it suitable for caving. Ground conditions in parts of the Live Oak section are such that the haulage level below the ore is maintained with difficulty and trouble has been experienced in drawing the ore, because of its soft character. Underground costs per ton are shown in part in Table I.

Ray Consolidated Copper, Ray, Ariz.—The ore of the Ray mines occurs as secondarily enriched chalcocite in a quartz-sericite schist. Secondary quartz, sericite, and pyrite occur throughout the schist. A number of fault systems running in different directions cut both ore and capping. These tend to prevent arching and doming of the ore and capping when drawing ore. The schistosity has no uniform strike or dip. Usually, the ore tends to break in slabs parallel to the schistosity. Sericite is abundant in certain sections. The ore is soft, and when damp tends to pack and hang up when drawn. At Ray, the orebody is 7,000 ft. long and 1,500 ft. wide. The ore varies from 40 to 400 ft. thick, the average being 120 ft. The capping varies from 40 to 600 ft. thick, averaging 250 ft. All permanent workings in ore must be timbered.

Table II—Detailed Costs Per Ton of Block-Caving Method, Miami Copper Company—1926-30

Data From Annual Reports

Item	1926	1927	1928	1929	1930	1926–30
Development	$0.100	$0.100	$0.100	$0.100	$0.100	$0.100
Stoping	0.118	0.146	0.156	0.163	0.174	0.151
Electric haulage	0.052	0.060	0.047	0.052	0.043	0.051
Pumping	0.002	0.002	0.002	0.002	0.001	0.002
General underground	0.011	0.012	0.011	0.011	0.009	0.011
Mine ventilation	0.017	0.017	0.014	0.015	0.013	0.015
Underground lighting	0.003	0.003	0.003	0.003	0.002	0.003
Hoisting No. 5 shaft	0.033	0.034	0.029	0.030	0.024	0.030
Engineering and sampling	0.012	0.014	0.014	0.015	0.010	0.013
Mine accident	0.003	0.010	0.012	0.011	0.006	0.008
Mine surface	0.020	0.018	0.019	0.021	0.014	0.018
Totals	$0.371	$0.416	$0.407	$0.423	$0.396	$0.402

The Ray method of caving was originally a combination of shrinkage stoping followed by undercutting the pillars between stopes. This method has been modified, but shrinkage is still used as the means by which undercutting is accomplished. At present four variations are in use, known as (1) hand-tramming method, (2) motor method, (3) modified motor method, and (4) sublevel-caving method. Selection depends

This article is one of a series on the cost of metal mining in the United States.

upon the thickness of the ore, its elevation above the haulage level, and other physical characteristics. Approximately 70 per cent of the tonnage is now being produced by the sublevel method. Costs per ton are shown in Table I.

Morenci Branch, Phelps Dodge, Morenci, Ariz.—Orebodies of the Humboldt mine occur in a quartz-monzonite porphyry which in many places has been completely altered to sericite, quartz, and chalcocite with its associated pyrite. Irregular fracturing occurs throughout the orebody, planes of weakness being from 2 to 18 in. apart. The orebodies have no definite walls, the boundaries being assay limits. Temporary workings generally stand without timber, but all permanent drifts and raises in ore must be timbered. The Humboldt ore is somewhat harder than that of the other porphyry mines in the Southwest. Capping consists of iron oxides and silica and does not tend to pack.

Morenci undercut block-caving method is similar in principle to other block-caving methods and varies from them only in operating details. The method uses an undercutting level and, 20 ft. below it, a grizzly level, and 55 ft. below the grizzly level, a haulage level. The panel mined is 112 ft. wide and runs the full length of the orebody. Costs are shown in Table I.

Ruth Mine, Nevada Consolidated Copper, Ruth, Nev.—The Ruth orebody occurs in a porphyry area entirely surrounded by sedimentary rocks. Copper occurs as chalcocite and chalcocite-coated pyrite with a little chalcopyrite. The orebody is 1,600 ft. long and 1,200 wide and has an average thickness of 120 ft. Capping averages 410 ft. thick. The ground under the orebody, including both the porphyry and the sedimentary rocks, is soft and heavy, and all openings require close timbering.

The first method employed was to take out narrow shrinkage stopes and cave the pillars between by undercutting. Pillar ore and shrinkage ore were drawn together. Owing to the soft ground underlying the ore, excessive weight developed in the haulage level. The branch-raise system was then installed. Undercutting and drawing are done from small finger raises at 12.5 ft. intervals in both directions. This change proved very successful. In 1924, the mine was producing approximately 2,700 tons per day. Costs per ton are shown in Table I.

Cost Factors—The four most important factors which determine the cost per ton by the block-caving method are: (1) height of lift; (2) breaking characteristics of the ore; (3) packing characteristics of the ore; and (4) strength of ore or waste below the ore being caved.

Of all factors affecting cost, the height of the ore column is most important. Since block caving has been introduced in the Western United States, the practice has been toward an ever-increasing height. Experience has shown that blocks over 300 ft. high can be most successfully caved with one undercutting. This reduces to a minimum the number of grizzly and undercutting levels and the haulage development work. The effect of the height of ore column is strikingly shown in Table III.

Table III—Mining Costs, Block Caving With Varying Thickness of Ore, Miami Copper Company (*a*)

Ft. of Ore	Cost per Ton	Difference per 25 Ft.
25	$1.56	.....
50	0.96	$0.60
75	0.73	0.23
100	0.62	0.11
125	0.55	0.07
150	0.50	0.05
175	0.46	0.04
225	0.40	0.02
250	0.38	0.02
275	0.36	0.02
300	0.35	0.01
325	0.34	0.01
350	0.33	0.01
375	0.32	0.005
400	0.32	0.005

(*a*) T. P. No. 314 A.I.M.E.

In addition to a high lift the ideal conditions are:

That ore or waste below the ore being caved be sufficiently strong to support the weight induced by caving operations, so that haulage and grizzly levels can be maintained without excessive repairs.

That the ore being caved does not contain excessive amounts of sericite or kaolin or other wet clayey material which will make it pack, chimney, and hang up.

That the ore be friable and that it break into small pieces when undercut.

Whenever the conditions mentioned are unfavorable, high costs result. The ore at Morenci is harder than in the other mines using caving, and much difficulty was had in drawing coarse ore. In all of the other mines, the spacing of grizzly bars was from 10 to 12 in., but at Morenci it was increased to 16 in. The output per grizzly man-shift with 10-in. spacing was 37 tons, whereas with a 16-in. spacing the output was increased to 120 tons. At first thought, this change appears almost self-evident and inconsequential. However, the change necessitated increasing the size of all chute mouths and extensive changes to the haulage, hoisting, and coarse-crushing equipment to handle the coarser material produced.

In 1929 at Inspiration the output per man-shift for the sulphide ore was 9.22 tons, and for the oxide ore 13.79 tons. This increase of 50 per cent in output was caused almost entirely by the lesser packing characteristics of the ore, and to the better ground conditions which existed in the haulage and grizzly levels below the orebody. Caving methods at both the Ray and Ruth mines have undergone changes better to adapt them to the caving of soft ore and to reduce the expense of the upkeep of workings below the ore.

Other Costs—The costs of mining by block caving already cited included five of the seven mines in the United States using this method. The two other mines are the Copper Queen, at Bisbee, Ariz., and the Old Dominion, at Globe, Ariz. Copper Queen costs have not been published, and Old Dominion costs, which are incomplete and partly estimated, are given in Table IV. In 1929, approximately 850 tons per day was produced by the method.

Table IV—Estimated Direct Mining Cost (*a*) Per Ton, Inclined Slice Method, 1929

Development	
Labor	$0.060
Timber	0.038
Shrinkage	0.021
Undercutting	0.136
Drawing at 40 tons per man-shift	0.130
Powder (development and bulldozing)	0.040
Repairs	0.050
Total	$0.475

(*a*) I.C. 6237.

Ohio Copper, Bingham Canyon, Utah—Mining literature unfortunately does not include complete itemized costs of caving as employed at the Ohio Copper Company's mine, the first of the Western mines to use the method. This discussion would be incomplete without reference to the fragmentary data on the subject.

Stevens Copper Handbook, Vol. 10, states that mining costs are said to have averaged 31c. per ton in 1910 and were reported as 27.7c. in August, 31.8c. in September, and 31.3c. in October, 1910. Total mining and milling costs were said to be 84c. per ton in February, 1911.

Clarence G. Bamberger, in *Engineering and Mining Journal*, April 6, 1912, gives the costs for October, 1911, as in Table V.

Table V—Ohio Copper Costs, October, 1911

Mining with delivery to bins, cents per ton	21.97
Mining, including development, cents per ton	28.06
Tons per man-shift underground	17.0
Tonnage mined per day	1,800
Tonnage mined October, 1911	56,311
Wages: Machine men	$3–$3.25
Timber men	$3.25
Muckers	$2.50–$2.75
Helpers	$2.75

F. Sommer Schmidt, in *Mining and Scientific Press*, March 6, 1915, says that 3,056,000 tons of ore was mined since 1909 at a cost under 25c. per ton for three years. For the past twelve months the cost was 24c. per ton, of which 2c. is for development. Extracted ore from drifts, crosscuts, and raises cost 79c. per ton and the caved ore cost 6.8c. per ton, giving an average of 22c. per ton for the total tonnage.

When one considers that caving at that time was in its infancy, costs as low as these are truly remarkable. The lower wages of that period partially contributed toward making such costs possible, but no doubt extremely favorable ground conditions were largely responsible.

Cost of Preparatory Work—One of

the largest items of the cost of caving is the preparatory work required before caving actually starts. These charges for the different mines are shown in Table VI.

Table VI—Cost of Preparatory Work Per Ton, Caving Methods

Miami, development	$0.100
Inspiration, development	0.211
Ray, estimated cost of preparatory work	0.146
Morenci, estimated cost of preparatory work	0.303
Ruth, development suspense, and construction	0.350
Old Dominion, partial cost of development	0.098

At the Ray mine, the total mining cost for 1914 to 1916 included 10c. per ton for prepaid development. In 1917 and 1918, this charge was 12.5c. per ton, and it increased to 15c. from 1919 to 1930 inclusive. In 1931, it was reduced to 6.5c. per ton.

At the Ruth mine, costs included a charge of 30 to 35c. per ton as development suspense and 5c. per ton as deferred construction.

Sublevel Caving—Sublevel caving and undercut block caving have so much in common that they are classified under the general heading of caving methods. The difference between the two methods is in operation rather than in the underlying principles involved. Both methods use the principle of inducing the breaking of ore by undercutting it so that it will cave. The greatest difference is in the ratio of the amount of ore recovered by undercutting to that recovered by caving. In caving operations in the West, the tendency has been to increase this ratio by using higher and higher lifts. The height of the first lift was only 30 ft. This was gradually increased until today blocks 300 ft. high are being successfully caved with one undercutting. The present undercut block-caving method is an evolutionary development of the older sublevel-caving method.

Sublevel caving in 1929 produced 7,000,000 tons, of which about 1,000,000 tons was produced by the Eureka-Asteroid mine, on the Gogebic Range, Michigan, and the Montreal mine, Montreal, Wis.

Eureka-Asteroid Mine, Gogebic Range, Michigan—The iron ore at the Eureka-Asteroid mine occurs in sedimentary rocks in troughs formed by narrow intersecting dikes normal to the dip of the sedimentary series. In this mine, the ore follows the beds, which dip from 55 to 75 deg. The ore is a soft hematite that requires the support of timber in all openings. The hanging wall is a cherty iron formation, which is fairly strong but caves when the ore is removed. Usually, the ore is drilled with auger steel and hand-held drills. A sublevel method of mining is used. Sublevel drifts are 25 ft. apart horizontally and 18 ft. vertically. Scrapers are used for moving ore to 65-deg. inclined, cribbed raises. Stoping is done at a contract price of 60 to 90c. per 4.5 ton car. Total costs per short ton are shown in Table VII.

Montreal Mine, Montreal, Wis.—The iron ore of the Montreal mine occurs in a series of sedimentary rocks under either of two conditions. One type occurs in 20-deg. pitching troughs between intrusive dikes and impervious quartzite and slate. The ore dips steeply with the beds. The other type of orebody is an irregular chimney orebody following faults to a great height above the dikes. These chimneys have a pitch of about 80 deg. The ore is medium soft and is usually drilled with auger drills, but occasionally a few hard ribs require the use of mounted hammer drills.

Two methods of mining are used: sublevel caving for the trough orebodies and sublevel stoping for the chimney orebodies. This selection is made because in the trough orebodies the hanging wall caves, whereas in the chimney orebodies the capping is strong and stands well. Total costs per short ton are shown in Table VII.

Table VII—Sublevel Caving Costs Per Short Ton

	Eureka-Asteroid, Gogebic Range, Mich., I.C. 6348	Montreal, Wis. I.C. 6369
Stoping		
Labor	$0.391	$0.333
Supervision	0.023	0.031
Explosives	0.061	0.060
Timber	0.061	0.054
Air, drills	0.060	0.018
Power	.....	.....
Miscellaneous	0.007	0.025
Total stoping	$0.603	$0.521
Development	0.282	0.226
Transportation	0.363	0.306
General	0.120	0.077
Miscellaneous	.....	0.062
Totals	$1.368	$1.192
Tonnage	450,000	500,000
Year	1929	1928
Explosives, lb.		
Stope	0.34	0.35
Total	0.63	0.53
Timber, bd. ft.		
Total	2.98	2.46
Tons per man-shift		
Underground	6.42	7.31
Total	5.00	6.72

Table VIII—Total Cost of Mining Per Ton by Block-Caving Method Except Depreciation and Taxes

From Annual Reports

Year	Miami	Inspiration	Ray (*d*)	Ruth(*e*)
1916	$1.12(*a*)	$0.62	$0.66	$1.41
1917	1.27	0.75	0.94	1.18
1918	1.36	0.73	1.58	1.33
1919	1.54(*b*)	0.77	1.55	1.35
1920	1.30	0.73	1.65	1.32
1921	1.15	0.81	1.42	1.21
1922	1.11	0.81	1.08	1.24
1923	1.23	0.70	1.21	1.45
1924	1.13	0.63	1.21	1.32
1925	0.85(*c*)	0.81	1.09	1.20
1926	0.37	0.76	0.96	1.36
1927	0.42	0.77	0.92	1.33
1928	0.41	0.74	0.84	1.42
1929	0.42	0.80	0.98	1.35
1930	0.40	0.76	0.85	1.34
Average 1916-30	$0.94	$0.75	$1.13	$1.32
Average 1926-30	0.40	0.77	0.91	1.36

(*a*) Production 3,000 tons daily from top-slice stopes and 3,000 tons from shrinkage and pillar caving.

(*b*) Top-slicing stopes and total production from undercut caving stopes began in 1920.

(*c*) The start of production from the low-grade orebody by block-caving high lifts.

(*d*) Ray costs include 10 to 15c. per ton for prepaid development. Cost of coarse crushing at the mine is not included. Cost of square-setting from 1917 to 1920 is not included.

(*e*) Ruth costs include 30-35c. per ton as development suspense and 5c. per ton for deferred construction.

Table VIII has been prepared to show the total cost per ton of mining by the block-caving method since 1916. These costs have been taken from the annual reports issued by the four most important users of the method. The costs embrace all items of expense, including prepaid development and exclude only depreciation and taxes.

Such small variations in the costs over the fifteen-year period of Inspiration and Ruth mines are surprising. The big reduction at Miami began in 1925, when the low-grade orebody was brought into production. A reduction in costs at Ray is also noted, beginning with 1926. In conclusion, the cost of mining by the block-caving method can be said to vary from $0.40 to $1.35 per ton.

Statistical data of undercut block caving are shown in Table IX immediately following.

Table IX—Statistical Data, Undercut Block-Caving Method

Mine	Reference	Explosives per Ton	Bd.Ft. per Ton	Tons per Underground Man-shift	Total Man-shift	Year
Miami	A.I.M.E. Vol. 72	0.33	3.6, 0.33 lin.	7.58	6.82	1922
Miami	A.I.M.E. T. P. 314	0.22	1.05	27.0	.....	1925-29
Inspiration	I. C. 6169	0.20	.....	13.1	.....	1928
Inspiration	Annual reports					
	(Oxide ore)	....	.....	.....	13.55	1926
	(Oxide ore)	....	.....	.....	12.45	1927
	(Oxide ore)	....	.....	.....	13.55	1928
	(Oxide ore)	....	.....	.....	13.79	1929
Ray	I. C. 6167	0.25	1.79	10.66	9.39	1927
		0.32	1.88	12.13	10.83	1928
Ruth	Annual reports	....	.....	10.39	7.53	1918
		....	.....	11.51	8.34	1919
		....	.....	12.27	8.99	1920
		....	.....	13.08	9.38	1921
		....	.....	9.66	.....	1924
		....	.....	11.80	.....	1925
Morenci	I. C. 6107	0.24	0.23	10.00	9.07	1927
		0.19	0.25	11.84	10.45	1928
Average (*a*)		0.25	1.47	11.17	10.32	.....

(*a*) Miami, 1925-29, not included in tons per man-shift underground. Including Miami, average is 12.7 tons.

Hoist Design and Construction

Lucien Eaton

HOISTS for shaft-sinking and for the operation of small mines have been discussed in a previous article, whereas this one will be confined to hoists for production shafts only. Exploratory shafts and those in the exploration stage of development do not need equipment as elaborate as do production shafts, but the calculations as to capacity and power, and the safety precautions necessary, are similar.

Hoisting is usually done with two skips or two cages in balance, or with a skip or cage balanced by a counterweight. Unbalanced hoisting is comparatively rare in new installations, except in flat inclines. The arrangement of a skip and cage in balance is not recommended, because it causes too much delay, one being unavailable while the other is in use. For the same reason, cages hung above skips are generally unsatisfactory.

If the power requirements of the hoist, whether it is driven by steam or electricity, are a large proportion of the total power available, variations in steam pressure or voltage occurring at the peak hoisting load will interfere with the satisfactory operation of the other power-consuming units. To equalize the load of steam hoists as far as possible, conical drums, or more frequently cylindro-conical drums, in which the central portion is cylindrical and the ends are conical, are sometimes used. These reduce the power requirements for starting and acceleration, but materially increase the cost of the drum. The addition of a tail rope, extending from the bottom of one skip around a sheave at the bottom of the shaft to the bottom of the other skip, equalizes the load still further. This practice is common in Europe, but has never found favor in the United States.

Other systems of hoisting, having as their object greater equalization of load and more rapid acceleration, have been developed, the most important being the Koepe and Whiting systems. The Koepe system, which is extensively employed in Germany, uses no drum; the rope passes instead over a large sheave, to which it is not attached, tractive effort being imparted to the rope solely through its friction in the groove. To develop the necessary friction, 180 deg. of contact is required, and the groove is lined with wood or "Ferrodo" fiber. In the Karlick sheave—made by Heckel, Saarbruecken, Germany—the sheave groove is filled with special clamps that grip the rope when it is under tension, and reduce to a minimum its tendency to slip. The moving parts of the Koepe hoist are light, and the power required for acceleration is relatively small. As the head sheaves are usually placed one above the other, and the hoist is on the center line of the hoisting compartments, the fleet angle is zero. The depth of the shaft does not affect the size of the sheave, except that under ordinary circumstances the load on the heavy side should not exceed 1.60 times that on the light side. This ratio is kept down by using a tail rope, which is sometimes heavier than the hoisting rope.

In spite of obvious advantages, however, the system has certain drawbacks that have militated against its adoption in the United States. Chief of these is the danger attendant upon slippage of the rope. Moreover, no adjustment in the length of the rope is possible, so that short pieces cannot be cut off from the ends, as is usually done at frequent intervals when a drum hoist is used. However, frictional wear on the rope is slight, and the use of an oversize rope insures long life. The system is best for hoisting from one level only, and the use of a tail rope is almost a necessity. If the rope breaks, at least one skip or cage will go to the bottom of the shaft, because enough friction will be developed by the rope in passing over the sheave to prevent the safety catches from working.

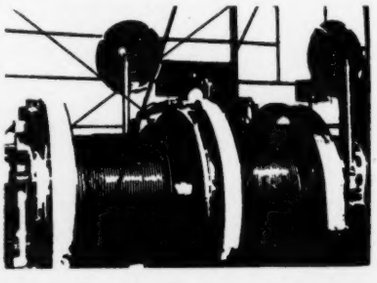
In the Whiting system the hoisting sheave is of smaller diameter than the Koepe sheave, and the requisite amount of contact for tractive effort is obtained by passing the rope around the sheave and an idler four or five times. The idler consists of three or four adjacent sheaves of the same size, loose on one shaft, and another sheave, mounted on a truck that travels on a short track. By changing the position of this movable sheave, the length of the rope can be adjusted at will. This overcomes one objection to the Koepe system, but introduces another; for, if friction develops between the adjacent idler sheaves, the strain on the rope and on the idler shaft becomes very large, and power consumption increases correspondingly. In another form the main idler is a single sheave on which there are separate, loose, grooved rings for the rope; these rings shift enough to equalize the tension on the rope.

Both the Koepe and Whiting systems reduce the fleet angle to a small figure; and, on account of the small weight of their moving parts, permit of rapid acceleration. However, few, if any, installations of the Whiting system have been made in the last 25 years, and the Koepe system is almost unknown in the United States.

Reels, in which flat ropes are wound upon themselves, also reduce the fleet angle almost to zero, require a low starting torque, and, being light in weight, permit rapid acceleration; but their rope expense is so great that it outweighs all advantages, and their use has been almost entirely abandoned.

The use of electricity for power has made possible an absorption in the electrical equipment of much of the variation in load; and for this reason the present tendency in design is toward straight cylindrical rather than conical drums, and also toward drums of smaller diameter and greater speed. A tendency is also in evidence at mines where loading is all done at one level toward automatic or pushbutton control, by means of which the hoist is started by the man who fills the skips, and its subsequent movement is controlled automatically. By the use of automatic control the acceleration and deceleration is always at the predetermined rate, peak loads are kept at a minimum, and the personal equation is eliminated. The first cost of automatic equipment is higher, but a substantial saving is effected in the cost for labor in operating.

The diameter of a drum should be not less than 60 times the diameter of the rope, and a larger factor will be better for the rope. If most of the hoisting is done from the bottom level, and if the rope is wound in a single layer, a single drum, keyed to the shaft, is a satisfactory design for a skip hoist. The same is true for cage and counterweight, no matter how many levels must be served. If the rope is wound in multiple layers, a division wall must be provided in the face of the single drum, or two drums must be used. Both drums may be keyed to the shaft. If, however, the tonnage to be hoisted is divided fairly equally between two or more levels, two drums are to be preferred, of which one is keyed to the shaft and the other is held by a clutch. Sometimes both drums are



This is one of a series of articles on mine equipment.

loose on the shaft and are equipped with clutches, but this involves an unnecessary additional expense. By this arrangement the length of the rope in use can be quickly adjusted so that hoisting from any level can be carried on without over-running. Hoisting capacity is thereby increased and power consumption reduced; whereas, if the drums are keyed to the shaft, hoisting capacity on the upper levels can never exceed that on the lowest level served by the skips. This latter consideration is unimportant if hoisting capacity is in excess of requirements, or if the greater part of the hoisting is done from the lower level. Having one drum loose on the shaft, however, is a great convenience when a new rope is put on or an old one is turned end for end or adjusted in length.

In unbalanced hoisting a single drum is used, which may be keyed to the shaft or be loose and equipped with clutch. The former arrangement is often preferable for an electric hoist and the latter for a steam hoist.

Large drums are usually cast in sections. The ends, including the flanges and brake rims, and the spiders are cast separately, and the drum face is cast in sections or is rolled from heavy plate. Steel, electrically welded and cast, is taking the place of cast iron to a large extent in drum construction, and its use is recommended. Grooves are usually cast in the face of the drum to direct the winding of the coils of rope. The rope should fit loosely in these grooves, and their distance apart, center to center, should be $\frac{1}{8}$ in. greater than the maximum diameter of the rope. If only one layer of rope is to be used, the grooves are cast in a spiral; if the rope is to be wound upon itself they are cast in circles normal to the axis of the drum, so that the coils in succeeding layers will lie smoothly upon each other. The rope must climb over the ridge between grooves once in each revolution, but this seems to have little injurious effect. In multiple-layer winding, if a drum without grooves is used, the rope must be wound in such a way as to make an exact number of complete turns. Otherwise it will be jammed between the last lap and the flange. Under any circumstance a certain amount of shock results when the rope climbs over the first layer and drops into the groove between two turns. At least three full turns of rope are usually left on the drum when the skip is in its lowest position. When both skip ropes are wound on the same drum at least two vacant grooves should be left between the two ropes. In calculating drum capacity under these conditions, therefore, at least eight grooves must be allowed in excess of those required for the maximum travel of the skip. The depth of the drum flanges should be at least 4 in. greater than the total thickness of the layers of rope.

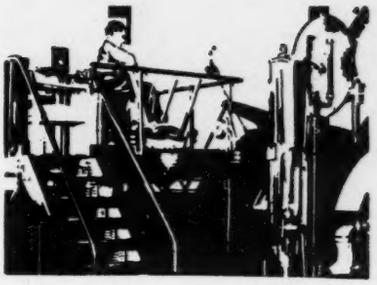
The question of fleet angle is important in relation to the width of the drum face and the distance between the drum and head sheave. By fleet angle is meant the maximum angle that the rope makes with a plane normal to the axis of the drum, and authorities seem to agree that it should not exceed 1 deg. 30 min. A large fleet angle causes excessive wear on rope and drum, and it should be avoided, even if multiple-layer winding must be adopted.

Methods used to obtain a small fleet angle vary radically, and the extremes of practice can be seen in the Lake Superior Iron Country and on the Witwatersrand, in South Africa. In the former district the rope is wound in a single layer on a drum of comparatively small diameter and wide face, and the fleet angle is kept low by erecting the hoist at some distance from the shaft, the empirical rule enjoining a distance of 50 ft. for every foot of rope-travel on either side of the center line of the drum. On the Rand the hoist is usually erected comparatively near the shaft, and the fleet angle is minimized by winding the rope in multiple layers on narrow-faced drums of relatively large diameter.

Clutches may be of the friction or jaw type. The former, of which there are two classes—band-clutches of the Lane type, and multiple-arm disk clutches—are preferred in the United States. In the Lane clutch a band, similar to a brake band, is tightened about a clutch ring at one end of the drum; in the disk clutch two annular rings, or disks, lined with asbestos fabric or blocks or similar material, are made to grip between them a corresponding steel ring attached to the drum. In both types the clutch is carried by a spider, keyed to the shaft. In South Africa and in parts of Europe a strong prejudice exists against friction clutches, and jaw clutches are required by law. The usual type of two- or three-jaw clutch does not allow close enough adjustment of the length of the hoisting rope for balanced hoisting, and in its stead a clutch made with multiple teeth, like interlocking internal and external gears, should be used. Jaw clutches are positive in action, but can be thrown in only when the hoist is at rest or moving slowly. On large hoists the clutches are operated mechanically by means of cylinders operated by compressed air or by oil under pressure.

On small hoists the usual practice is to use band brakes, but the majority of large hoists are equipped with post brakes. The latter are much more expensive, but require less movement between the on and off positions, have the same holding power for both directions of rotation of the drum, and always carry the load on the stationary point of attachment to the frame.

If a band brake is used, and hoisting is not done in balance, a continuous band with one end attached to the frame is best. The band extends around the drum in the direction of rotation, when lowering. By this arangement the brake is self-tightening; it can be



operated by hand, if desired, on hoists up to 500 hp. The band is kept from dragging on the drum, when released, by tension springs radially attached at intervals of 45 to 60 deg. on the upper side. If, however, hoisting is in balance, the brake band should be in halves, and is attached to the frame by a flexible connection at the junction of the two parts. The loose ends are tightened or loosened equally by the operating mechanism. When released, the upper half is kept away from the drum by radial tension springs and the lower half by gravity. By this arrangement, whichever way the drum may be rotating, one half of the brake is working with the load and the other half against it, and the braking effect is the same in both directions, as it is with post brakes. The arc of contact, however, is one-third larger than with post brakes.

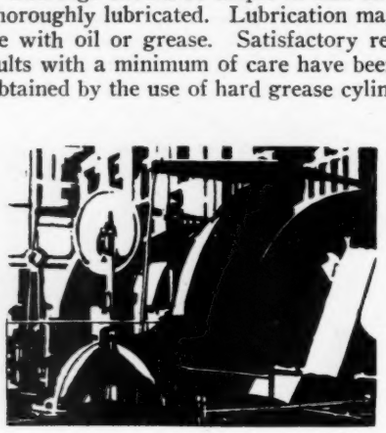
Post brakes are made of two rigid beams or posts, curved to make contact with the drum on an arc of 100 deg. They are attached at the middle to the frame of the hoist by a flexible connection, and are tightened or loosened by parallel movements of both ends. In the smaller hoists the posts are steel castings; in the larger they are built up of plate and structural shapes.

Band brakes that are made in halves and post brakes, except on small hoists, are actuated by power. The usual practice is to tighten the brake by a weight attached to a lever, and to release it by raising this lever with a compressed-air or hydraulic cylinder. Power is supplied by a small independent air compressor and receiver, or by a pump and hydraulic accumulator. The weight is adjusted to hold the drum under the maximum unbalanced load. Brake bands or posts are lined with wood blocks, usually basswood, cut to the proper shape and bolted in place, or with special blocks made of asbestos or "Brake-Blok," a compound made especially for this service by American Brake Materials, a subsidiary of the American Brake Shoe & Foundry. Asbestos brake blocks are sold by Johns-Manville, made up in sizes to suit. "Brake-Blok" and asbestos linings have advantages over wood in that they retain to a large degree their gripping power, even though oil gets on the brake rim, and have a long life.

Every large hoist, in addition to the service brake, should have an emergency brake operated by automatic controller. When large post brakes are used, the automatic controller is usually attached

to the service brake, but a separate brake is better. The service brake is generally applied to a brake rim, which is a part of the end-section of the drum and has the same diameter as the drum flanges. The emergency brake is usually a duplicate of the service brake, but is at the opposite end of the drum; or, if the power is applied through a set of reduction gears, it may work on a brake drum on the pinion shaft. Either design is satisfactory, but the former is preferred, and it has more general application.

Bearings should be ample in size and thoroughly lubricated. Lubrication may be with oil or grease. Satisfactory results with a minimum of care have been obtained by the use of hard grease cylin-



ders, shaped like candles, which are pressed down on top of the shaft by spring feeders. A double-drum hoist should always have a bearing between the drums; and, if the gear is keyed to the shaft, a bearing should be provided on each side of it.

On geared hoists the coupling between the pinion and the motor is usually of the flexible type. A recent development is a hydraulic coupling produced by the Metropolitan-Vickers for the Rhodesian mines.

Large high-speed hoists are usually driven by direct-current motors, mounted directly on the drum shaft or geared. The Ward Leonard system of control is used, in which the (direct) current is supplied by a motor-generator set and the speed and direction of the hoist are regulated by the amount and direction of the excitation in the field of the generator. This system gives almost perfect control with small resistance losses, and is therefore especially advantageous for sub-normal speeds. As a shunt-wound motor is used, overbalancing of the load regenerates current and causes automatic retardation. Automatic control is easily applied, as the speed of the drum depends on the amount of excitation without regard to the load. The main disadvantages are the high first cost and the idling loss of the motor-generator set. Ward Leonard control is often combined with an Ilgner motor-generator set, in which a heavy flywheel is mounted on the same shaft with the generator, exciter, and motor. The motor is of the slip-ring type, arranged so that when an abnormal load is thrown on the generator the motor will lag, and the peak will be carried by the flywheel. This irons out the peaks on the supply circuit, but the power consumption for idling is rather high. If the alternating-current power supply is large, the flywheel may be omitted, and a motor with high pull-out torque is used, of the slip-ring or synchronous type. Almost complete elimination of peak loads can be obtained by the use of a synchronous motor without flywheel, and this type of drive seems to be gaining in favor.

To minimize capital expense, Ward Leonard control is often used on the skip hoist only, and the cage hoist is driven by an alternating-current motor. This is an economical arrangement, because the idle periods are much longer with the cage than with the skips, and the motor has no idling expense.

In alternating-current motor drives, wound-rotor, slip-ring, induction motors are geared to the drums, and best results are obtained through single-reduction, double-helical, cut-steel gears. The gear reduction may be as great as 15 to 1, and quiet operation and long life are given by reductions of 10 or 11 to 1. To use gears of ample width is good policy, and a width at least 50 per cent greater than that theoretically required should be insisted upon. Gears should be completely inclosed and thoroughly lubricated. The gear may be bolted directly to the drum, or it may be keyed to the shaft. The pinion and its shaft should be in one piece, mounted with a bearing on each side.

A mine may have several years of life on its upper levels, where a fairly low hoisting speed will give sufficient capacity. If this condition is known in advance, a considerable saving can be made by equipping the hoist with one motor at first and adding another later. If alternating-current motors are used, the increase in speed is obtained by the use of larger pinions. The second motor is mounted on the opposite side of the gear, and the two motors are run in parallel, the only changes in control necessary being the addition of a few more contactors and resistance grids. If the change is planned in advance, the bed-plate can be prepared for the new position of the bearings. A Ward Leonard direct-current motor drive will require a duplicate motor-generator set for the second motor.

The hoisting system used for passenger elevators has in a few instances been successfully applied to cages. In some installations a hoist of the ordinary type is used, but at Morenci, Ariz., a regular elevator hoist with multiple ropes is installed on top of the headframe. The hoist is controlled from the cage through flexible cable, one end of which is attached to the bottom of the cage and the other to the timbers of the shaft, halfway down. The limit of depth for this type of hoist at present seems to be about 1,000 ft., because at that depth the load on the flexible cable approaches its tensile strength. Operation is safeguarded by numerous electrical devices, and apparently only a short step is involved in progressing from this type of control to the automatic system used on many of our new high-speed passenger elevators.

Automatically controlled hoists are started by the man who fills the skips. He pushes a button or pulls a cord, which closes a relay, which in turn starts a small motor attached to a drum controller. This motor accelerates the hoist, and is automatically cut out when the hoist is up to speed, and cannot be started again until the controller is reversed. Deceleration is accomplished by a cam, driven by a shaft geared to the drum of the hoist, which pushes the controller back at a predetermined rate. When the skip has reached the proper point in the dump, it shuts off the current, applies the brake, and reverses the controller. By this type of control the operators' wages are eliminated, and the hoisting cycle is shortened by the amount of time that would be lost in signaling.

Every large hoist that is manually operated should be equipped with a Lilly hoist controller or its equivalent. This controller will shut off the current and apply the emergency brake if the speed exceeds a predetermined rate, if the operator does not decelerate in time, or if he hoists too far. By ringing a bell it gives warning of the proper point to start decelerating at each end of the trip. The cost of installation is about $1,500.

Limit switches which shut off the current and trip the emergency brake if the skip travels too far in either direction may also be installed in the headframe and down in the shaft, or on the indicator dial.

Various devices have been used to indicate the position of the skip or cage in the shaft. The most popular design is a large dial equipped with a hand, like that of a clock, driven directly from the drum of the hoist through gears or a chain and sprocket. The indicator should be set in such a position that the hoisting engineer can see it and the drum at the same time without turning his head. No windows or lights should be behind it, the light from which might cause eye strain. At night the indicator should be illuminated by a floodlight, and in addition a kerosene lamp should be kept lit in such a position that by its light the indicator can still be seen if the electrical power circuit fails.

Although knockers or pull bells actuated mechanically by a bell cord are used as auxiliary means of signaling, and should always be provided, hoisting signals are almost universally given by electricity. Three systems are in vogue: (1) Single-stroke or vibrating bells operated by direct current; (2) vibrating bells operated by alternating current; (3) loud-speakers operated by induction.

Direct-current vibrating bells should be 6 to 8 in. diameter; single-stroke bells should be at least 1 in. larger, and may be as large as 14 in. Current may be furnished by a bank of wet cells, by storage batteries, or by a small motor-generator set. The latter requires the least attention and gives most satisfactory service. The voltage should be from 60 to 110.

Vibrating bells for alternating current are of the Faraday type, and work satisfactorily at 110 volts. Current for the bells may be taken from the same source as the lighting circuit, or may be transformed from the general power circuit, but in either case it should have its own separate cable, so that trouble on the lighting circuit will not put the bells out of commission. The bells for the cage may be of the same type as those used for the skips, but the tone should be different, so that they may not be confused in the engine house. Cage signals should be repeated from the engine house when men are being handled; this can be easily arranged by using a three-wire circuit and placing a bell at each level.

Satisfactory ringer boxes have been developed in recent years, which are waterproof and foolproof. The wires enter the box from below; contact is made by pulling down a plunger that fills the gap between the poles. The pull is against a coil spring, and a collar on the plunger prevents it from being pulled too far and relieves the contacts of all strain.

A recent development in signals for hoisting is an induction system, operating on the same principles as a radio, a description of which has been published in this journal and by the U. S. Bureau of Mines. This system was originated by Mr. L. D. Stewart, electrical engineer for Oglebay, Norton & Company at Ironwood, Mich., for use at the Montreal nient, and parallel with them and as close down the shaft as far apart as convenient, and parallel with them and as close to them as is practicable are two aërial wires on the cage. By means of loud-speakers and a telephone, signals can be given from the cage, and telephonic communication in both directions between the cage and the engine house can be carried on while the cage is in motion or at rest. This system has been in use only a short time. Although it appears to possess certain advantages, it has not yet established itself in general practice. The ability to telephone directly to and from the cage is desirable, but it can be done in other ways. Cages operated like passenger elevators usually have a telephone circuit in the same cable as the control.

(To be continued)

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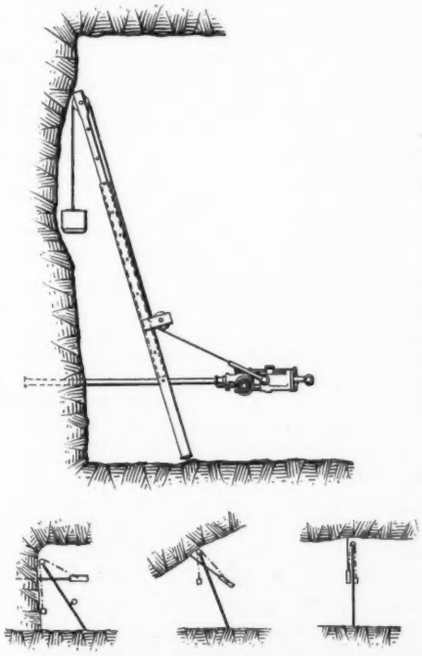
Counterweighted Drill Support

OUTPUT of miners using jackhammers in drilling horizontal, inclined, or vertical holes can be greatly increased by using a counterweighted drill support of the type shown in the accompanying illustration, writes Gaston Lejard, consulting engineer, Paris, France. This apparatus consists of a rod of adjustable length, with a pulley at the top. Weight of the drill is balanced by a counterweight suspended from a cable that passes over the pulley. The rod is made of two pieces of steel tubing, one sliding within the other. Adjustment of the rod to any desired length is obtained by inserting a steel pin through pairs of opposing holes drilled at regular intervals in each of the two tubes.



As indicated in the several sketches in the illustration, the resultant of the weight of the drill and the force in the cable is a thrust approximately in the direction of the hole being drilled. Under these circumstances the miner is relieved of the weight and vibration of the drill. Little effort is required to direct the drill, and, after the drill hole has reached a foot in depth, the drill, in many instances, may be left to operate by itself until the final depth of the hole is reached or until the drill bit in use must be changed. The apparatus makes possible the use of heavy drills of the jackhammer type. Although the arrangement is not applicable in all drilling operations, a marked increase in output is assured in those instances where it can be utilized.

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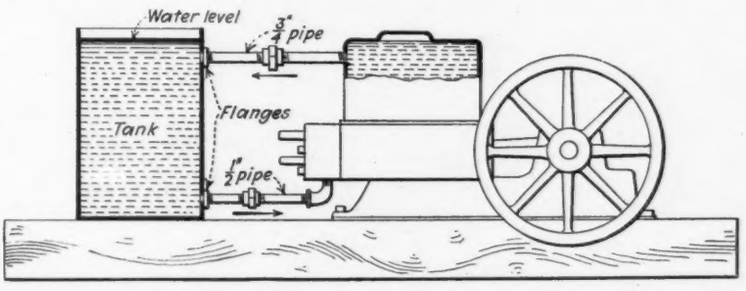
Cooling Small Gasoline Engines

IF A SMALL gasoline engine is in continuous service, such as when operating a mine blower, the water jacket is rarely found to hold sufficient water to prevent constant boiling, comments Evan W. Baker, of Winkelman, Ariz. Addition of an auxiliary water tank, properly piped, will overcome the necessity of adding cooling water many times each shift. The thermosiphon action between tank and engine prevents boiling in the engine jacket, and avoids the danger of the machine running dry, with resultant damage to cylinder walls and piston.

A wooden barrel, or steel oil drum, from which one end has been removed, makes a satisfactory tank. Holes are drilled in the side of the tank at the proper level, and half of a flange union is bolted over each. Heavy rubber gaskets should be used between tank and union. The lower circulating pipe may be connected to the engine by replacing the drain plug with a street elbow of the same size. A union and proper lengths of pipe complete the connection. For the upper connection it is necessary to drill and tap a hole in the water jacket of the engine about 2 in. below the water level. A short pipe is screwed into it, and then a connection made to the tank by means of a union and another piece of pipe. After filling the tank with water, the system is ready for operation. Circulation is in the direction of the arrows shown in the accompanying sketch. The water level in the tank should always be kept above the upper pipe.



Osmiridium in Tasmania

Harold S. Elford

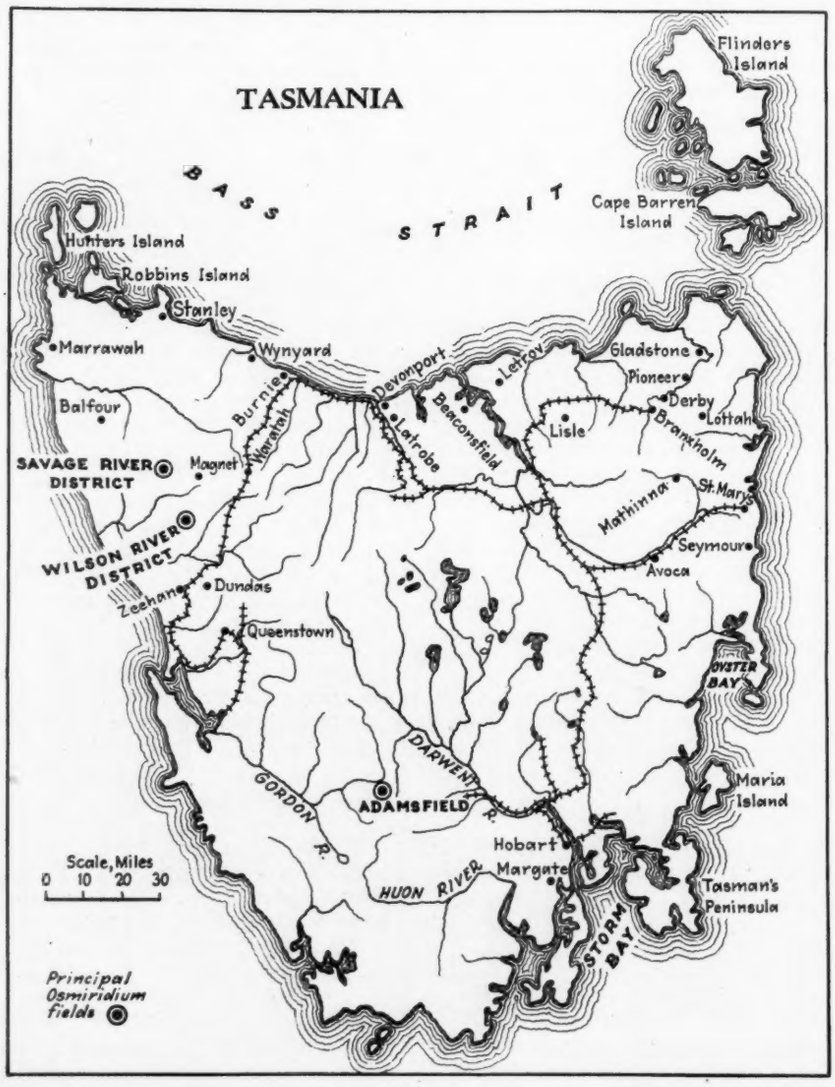
OSMIRIDIUM is a naturally occurring alloy of the two rare metals osmium and iridium, together with small and variable percentages of ruthenium, platinum, and rhodium. The chemical composition of osmiridium from the numerous occurrences in Tasmania and also within any one area differs widely. Adamsfield is the most important producing district in Tasmania, and an average of 31 complete analyses (Tasmanian Department of Mines, Geological Survey Bulletin No. 39, 1929, by P. B. Nye) of osmiridium, obtained from widely scattered alluvial workings, gave: osmium, 45.5 per cent; iridium, 41.7 per cent; ruthenium, 6.4 per cent; platinum, 1.10 per cent; rhodium, 0.30 per cent.

Nozzle at work sluicing shallow alluvial

Mineralogical classification of osmiridium is dependent upon the proportions of the two principal constituent metals, osmium and iridium. When these occur in approximately equal percentages, as at Adamsfield, the name nevyanskite is given; with a fall in the percentage of iridium to 30, the variety is termed siserskite. Nevyanskite, tin-like in color, has a specific gravity of 19. Siserskite has also been found in Tasmania, particularly at Mount Stewart, in the Savage River district. This mineral, which has a specific gravity of 15.5, is steel-gray in color. Osmiridium is usually found in small rounded grains, which accounts for the use of Tasmanian osmiridium in the tipping (hardening) of fountain-pen nibs.

Numerous small nuggets have been found on many of the fields; the largest recorded was obtained in the Savage River district and weighed 2½ oz. The only mineral of economic importance associated with osmiridium is gold, though native platinum and diamond have been found. Chromite, topaz, zircon, ilmenite, magnetite, and pyrite occur as accessory minerals.

Important quantities of osmiridium were first found in Tasmania in 1910 at the Savage River field, to the northwest of the famous but now idle Mount Bischoff tin mine. Production from the field in 1910 was only 120 oz., valued at less than £4/10/- per ounce. Interest was stimulated during subsequent years, production reaching 1,019 oz. in 1914, with a rise in price to £10 per ounce. The peak production from the Savage River district was in 1920, when 2,009 oz. yielded £77,000. Since then the output has gradually declined. Concurrently with the development of the Savage River area, attention was given to the Wilson River district, near to the tin field of Renison Bell, but production from this field has now almost ceased. In May, 1925, a remarkably rich area was located in the trackless, heavily timbered country along the Adams River, at the head of the Florentine Valley. A rush to the field followed, and despite the hardships of



Annual Output, Value, and Price of Osmiridium Won in Tasmania, 1910-31

Year	Quantity, Oz. Fine	Total Value, £	Value per Ounce, £
1910	120	530	4.4
1911	271.88	1,888	6.9
1912	778.77	5,742	7.4
1913	1,261.65	12,016	9.5
1914	1,018.83	10,076	9.9
1915	247.048	1,581	6.4
1916	222.150	1,899	8.5
1917	332.079	4,898	14.7
1918	1,606.743	44,833	27.9
1919	1,669.715	39,614	23.8
1920	2,009.196	77,114	38.4
1921	1,750.655	42,935	24.5
1922	1,173.924	35,512	30.3
1923	673.423	19,642	29.1
1924	364.805	10,617	29.1
1925	3,365.543	103,570	30.8
1926	3,172.5	61,908	19.5
1927	632.687	7,456	11.8
1928	1,627.186	42,458	26.1
1929	1,360	30,624	22.5
1930	952.7	16,235	17.0
1931	1,279.7	18,028	14.1
	25,891.2	£589,176	

transport and habitation, over 1,000 prospectors were encamped on the field by the end of 1925, and the whole of the island became thrilled with stories of rich discoveries. Production for the three months ended December, 1925, was 2,258 oz., valued at £68,000. Uncontrolled marketing of this large quantity brought a precipitous fall in price, and the number of prospectors decreased to less than 100. During the last few years, the number has increased to 200, and the output of the field has been comparatively steady at 1,000 oz. annually. Of the total Tasmanian production, the output from Adamsfield from 1925 to 1931 is 10,903 oz., valued at £235,300. This area, the only important producing field in Tasmania today, is 56 miles northwest of Hobart, the capital of the island. Access is by railway to Fitzgerald, 54 miles, the remainder of the journey being by saddle horse. Supplies to the field are brought in by pack horse, and are consequently expensive. The district is typical of other mining fields on the west coast of Tasmania—rugged, mountainous, and heavily timbered.

As in the Savage and Wilson River fields, Adamsfield osmiridium is genetically associated with basic and ultrabasic rocks, and occurs principally in the creeks that fall away from a prominent and persistent serpentine dike classed as of Devonian age. (Tasmanian Department of Mines, Geological Survey Bulletin No. 39, 1929, by P. B. Nye.) This dike has a maximum width of 5,000 ft. and is flanked on the west by sandstone and shales, and on the east by quartzite and conglomerate. Exposures show the serpentine as whitish green in color, a fact that permits outcrops to be seen from a considerable distance. Freshly exposed serpentine, however, is typically dark green and massive.

Methods of recovering osmiridium from alluvial wash are similar to those long used in winning alluvial gold. Panning is used for initial testing and for treating small quantities of particularly rich wash. A cradle may be employed when the water supply is limited, as is frequently the condition at Adamsfield, despite a rainfall of 70 in. per annum. When a stream of running water is available, the wash, which varies from 6 to 72 in. thick, is transported and broken up on a screen of steel plate having holes of ½ in. diameter. Coarse material remaining on the screen is rejected and the fine material passes along a wooden sluice box, 10 ft. long, in which the heavy osmiridium readily collects in riffles or on blanket. When water can be stored in sufficient quantity, or creeks on higher ground can be tapped, hydraulic sluicing is employed, the fine material being carried down to a sluice box. Frequently the osmiridium lies embedded on a clay (pug) bottom, in which case the clay is broken down in a puddling machine of the type shown herewith.

Operations to date carried on by individuals or groups of miners have been on a small scale. The grade of wash that can be worked profitably is dependent upon the price of osmiridium, which has varied widely.

On the principal osmiridium fields, considerable attention has been devoted to the search for lode formations carrying sufficient osmiridium, *in situ*, to justify treatment. On all fields but Adamsfield the search has been fruitless. During 1930, a true lode, carrying primary disseminated osmiridium *in situ*, was discovered, and has since been worked profitably by three claim holders. The lode, which has been proved for 700 ft., is 8 to 10 ft. wide, strikes northerly, and dips steeply to the west. The lode material consists of foliated serpentine lying between two well-defined walls; the country rock is dark-green serpentine. The osmiridium is confined to veins, up to 18 in. wide, which occur throughout the lode. Followed south, the lode carries little osmiridium; to the north it connects with slate country. So far, the lode has been worked to a depth of 60 ft. The decomposed lode material, brought to the surface in a small bucket hauled by whip and horse, is broken down by water pressure, and the osmiridium is recovered in a sluice box.

During the present depression, the price of osmiridium has fallen to £10 per ounce, at which figure most of the miners, of which there are 200 at Adamsfield, 40 at Savage River and 20 at Wilson River, are barely meeting living expenses. Production is being maintained at 250 oz. per quarter from Adamsfield, and a total of 50 oz. from the two other fields. The immediate future of the industry is almost wholly governed by the price of osmiridium, and substantial improvement in this respect would bring about increase in production. On all the fields, however, the rich shallow alluvial deposits have been exhausted, so that the real future of the industry is dependent on establishing cheap methods of winning osmiridium from the known extensive areas of low-grade and deeper alluvial, or on locating new fields.

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View along the open cut of the osmiridium lode at Adamsfield

Horse-operated puddling machine used for disintegrating clayey wash

Drilling With a Detachable Bit

A. H. Hubbell
Associate Editor

MUCH TIME might be spent in debating as to the reason why the detachable type of bit for rock drills has not found more extensive application in the mining and contracting fields. With respect to mining work, the fundamental idea is admittedly excellent. Why move tons of steel daily into and out of a mine, distributing it when sharp and collecting it again when dull, a few pieces at a time from widely scattered working places, that may also be hard to reach, when the task would be so much easier if only the bits plus a small part of the rods had to be transported? In a large mine the amount of steel that must thus be kept in daily circulation between drilling face and shop may be very large. The Anaconda Copper Mining Company found that a year's operation of one of its mines having 85 working places called for an installation of no less than 61,479 lb. An even weightier reason for making a bit detachable, in the opinion of E. R. Borcherdt, research engineer for the Anaconda company, is to be found in the elimination of time losses incurred by drillers whenever certain lengths or gages of drill steel are lacking, a not unusual circumstance attending the use of ordinary sharpened steel. As for contracting work, the availability of an efficient detachable bit at a reasonable price should prove an advantage to the contractor, by making it unnecessary for him to maintain on the job a sharpening shop, which in many instances would have to be moved frequently from place to place. The principal reasons in favor of detachability being therefore sound and eminently attractive, one may ask why have many decades rolled around since the first attempt to accomplish this result was made, without the principle having been applied more extensively. Whatever be the answer, a survey discloses some indications of a pending change.

That the detachable bit is in actual service today is of course well known. Anaconda's experience with the Hawkesworth bit at Butte and its final adoption of it have been duly recorded by the late C. L. Berrien in the *Transactions* (T.P. No. 274) of the American Institute of Mining and Metallurgical Engineers. The Granby Consolidated Mining, Smelting & Power Company, at Anyox, B. C., makes limited use of a bit of its own devising, which was described in these pages by W. J. Coulter in the issues of *Engineering and Mining Journal* of Dec. 4, 1926, and March 28, 1925.

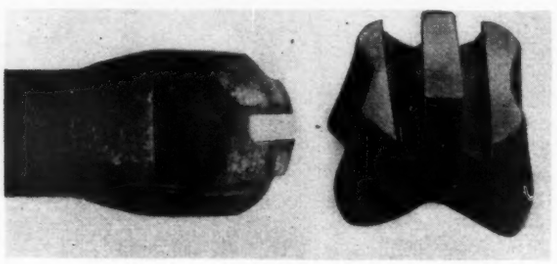

Fig. 1—A Hawkesworth bit and a shank, milled and ready for use. The bit is cruciform in type, the cutting edges being concealed from view.

These installations can be cited as proof of the practicability of the principle of detachability, if question is raised by anyone unfamiliar with these developments. Detachability being practicable in basic principle, a variety of practicable designs of detachable bits may reasonably be anticipated.

Bits of two designs have been mentioned. Aside from them it is difficult at the moment to point to an outstanding example where a large company has adopted a detachable bit for its routine drilling for breaking ground. This circumstance is nevertheless misleading. In the first place, an appreciable number of mining and quarrying companies and drilling contractors in the United States and Canada are either using detachable bits (other than the two already mentioned) on a modest scale for their routine work or are testing small lots of bits to determine their performance. These bits have been supplied by one or another of four manufacturers, of varying capacity, in the United States, all of whom are already in production. A fifth company is engaged in testing its product. Likewise, in Canada a promising bit has been devised and is being tried out by several large metal-mining companies there. Possibly still other bits have been developed under conditions so obscure that news of them has escaped attention. Models marking intermediate stages in the development of the bits referred to (and they are many) are not considered in this article.

The foregoing will serve to show that the detachable bit is at least a live subject and that the mining and quarrying industries may reasonably expect to see a greater use of it fairly soon. The extent to which it is likely to supplant ordinary sharpened steel is something that only the future can reveal. As further evidence of the trend, the fact may be stated that the Hawkesworth Drill Company, confident that the performance of its bit in the mines at Butte can be duplicated elsewhere, is merely awaiting more favorable conditions to attempt to service mines in other districts.

In this article space will be devoted to brief descriptions of the bits already available. No attempt will be made to criticise them or to compare them, because of possible unfairness and obvious pitfalls in such a course. Time will show whether or not any particular principle of design is best for drilling. Trial under actual operating conditions and in varying kinds of rock by many different users will eventually demonstrate, in a way not to be concealed, whether or not a bit is widely practical and the claims made for it justified. The development of several efficient bits differing markedly in design is possible.

Thus far the only advantages ascribed to making a rock-drill bit detachable from the shank or rod have been that of easing the work of distributing drill steel and of lessening time losses. There are other material advantages.

1. Steel of different analyses, if desired, can be used in bit and rod; for the bit, a steel made primarily for resistance to wear and impact; for the rod, one that will best resist fatigue and impact. With steels of such analyses available, a bit can conceivably be provided that will cut rock faster and stay sharp longer than a bit on ordinary steel; likewise, a rod that will have a longer life. In contrast, the analysis of steel used for making ordinary drill steel must be a compromise permitting the best all-around resistance to wear, fatigue, and impact.

2. Greater uniformity in the quality of a given bit is possible where the bit is made in a central shop than where the production or sharpening is scattered among many shops. This is just as true of detachable bits purchased from some bit manufacturer as it is of ordinary bits sharpened by a mining company having a central shop serving several mines. The advantage with respect to quality grows out of centralization of production (or sharpening), because such centralization makes economically possible better supervision and the employment of technical skill. This superiority will express itself, among other ways, in the provision and use of means of controlling more precisely the heat of forging and heat-treating.

Electric furnaces, which permit close control, are being used today to some extent in the production of both detachable bits and ordinary sharpened steel, but in the latter application in only a small proportion of the total number of mine shops. In many of these shops, especially at small mines, inferior bits are only too often turned out, thanks to dependence on the blacksmith's eye as a means of appraising temperature. This famous eye at its best is no substitute for a pyrometer. Bits, if forged at too high a temperature, are likely to break, or, if the temperature is too low, to crack and chip. The bits, moreover, may be poorly made aside from the quality of the steel. Carelessness in permitting sharpener dies to become dirty or in need of repair may result in bits being turned out under gage, or, by reason of lack of skill, they may be forged off center or may not be square with the rod. In the adoption of detachable bits, the advantage of uniformity will be at a maximum when the bit is to be used but once. A further advantage of centralizing production, as well as of increasing the scale of production, is the resultant lower cost per bit.

3. The decrease in quantity of steel required to equip a mine with detachable bits makes the necessary investment in steel smaller. Against the cost of the bit must be placed the total cost of resharpening an ordinary or standard bit, including interest and depreciation on the equipment used—this aside from any saving in other respects.

4. Depending on the particular detachable bit considered, the investment in shop equipment may be lessened compared with that required for resharpening and heat-treating ordinary steel. Where the bit is not to be used more than once, this investment may be reduced to small proportions. Only under rare conditions will it disappear entirely, because with any type of bit some equipment will be necessary for an occasional reshanking or reconditioning of the rods. In cases where the mine, quarry or drilling job is near the plant manufacturing the detachable bit used, the rods may possibly be shipped to the latter for the occasional reshanking or reconditioning required. This circumstance will be unusual, however.

In case a mining company undertakes to make its own detachable bits, it must of course acquire the necessary plant. Where detachable bits, once adopted, are to be resharpened or reground, the advantage of such resharpening lies in the saving made as compared with a cost of providing a new detachable bit. In this instance, the fact must be considered that with each reforging or heat-treatment the quality of the steel so treated diminishes.

5. If the detachable bit, as bought, is superior to the resharpened steel, or standard bit, turned out in the mine shop, increased drilling speed should result. The greater ease of handling detachable bits should also tend to speed up drilling.

6. The extent to which the use of detachable bits simplifies underground work should be reflected in a proportionate easing of the task of supervision.

When first considering the detachable bit and the possibility of applying it in the drilling work under his supervision, the average mine or quarry official is likely to have certain prejudices or fixed ideas which may act as obstacles to thought in comparing the proposed method of drilling with the existing one. First, he probably will think that his drill sharpening costs so little that a change is hardly worth while. This opinion is most likely to prevail where steel distribution is not a problem. Second, the change may seem to involve the scrapping of the steel and shop equipment on hand, and to possess uncertainties that in the end may rob it of its alleged advantages.

As to sharpening costs, the manufacturer of one detachable bit declares he has had great difficulty in getting mining companies which he has approached to determine their true costs for sharpening steel. Many of them, he says, surely have costs much higher than those they claim. Possibly this manufacturer is wrong in some instances. For neither side, however, is there any profit in self-deception. To ascertain the cost of sharpening should be a relatively simple matter, in contrast to that of steel distribution, which, if all indirect charges are included, may be rather difficult to calculate. The total cost of sharpening (that is, providing a resharpened bit fit for service) should include the costs of labor, fuel (or electric power), miscellaneous supplies and air used in making dull, bent or plugged steel fit for service again, plus liability insurance, plus interest and depreciation on the shop equipment, plus the first cost of steel and the interest on it, plus a proper charge for supervision. The cost of sharpening will depend on the number of steels sharpened. To sharpen and temper 97,791 steels at the Sullivan mine at Kimberley, B. C., in the first six months of 1932, cost the Consolidated Mining & Smelting Company of Canada 8.05c. each, this including labor and material for operation and repairs, and also electric power. In the corresponding period of 1929, when oil furnaces were being used and the shop was, moreover, much busier, 188,689 pieces of steel were sharpened and tempered at a cost of 7.62c. each for the corresponding items (the cost of oil being substituted for that of electric power). If the amount of steel sharpened is much larger than the figures cited, a correspondingly lower cost is to be obtained. For example, the Real del Monte company, in Pachuca, Mexico, in six months of 1929 sharpened (straigl ned, otherwise treated) 1,102,710 steels at the rate of 7,400 per day at a cost of 2.91c. per bit, which covered "labor, supplies, machine shop (dies, dollies) and power." In this shop electric furnaces were used. All of the items cited earlier in this paragraph as properly chargeable to sharpening have seemingly not been included in estimating the costs given in the three cases mentioned. At a small mine where five or six hundred pieces of steel are sharpened daily and the work is done none too efficiently, the cost of sharpening ordinary drill steel, exclusive of distribution and indirect charges, may be as high as 10c. per bit.

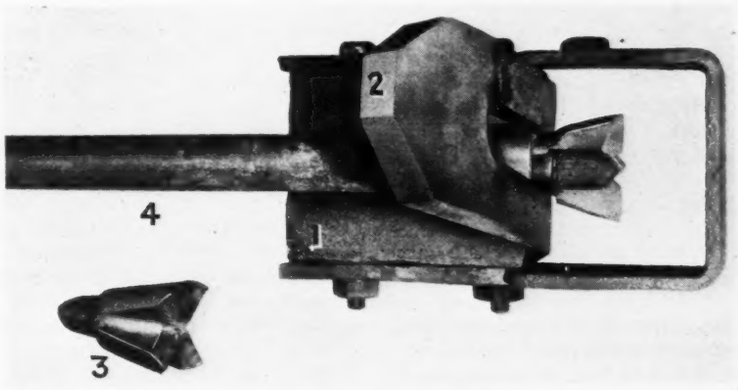

Fig. 2—Granby bit and rod, with bit-removing device. (1), Cast-iron block weighing approximately 30 lb. and shaped to receive socketed end of rod. (2), Bifurcated steel wedge, which is driven between shoulder of bit and that of rod to remove bit. (3), Detachable cross-type bit. (4), Hollow drill-steel rod.

Two items that may be slighted in estimating sharpening costs are the proportionate part of total air used in sharpening and drilling that is chargeable to sharpening, and the amount of steel consumed per bit resharpened. Unless the air taken by the sharpener is specially metered, the quantity used may be underestimated. The following data have been collected especially for this article: Actual test made by one

large iron-mining company in the eastern United States show that 4.56 per cent of the total air is chargeable to sharpening. This does not include air used for the forges and tempering machines which is provided by pressure blowers. A large zinc-lead mine in Canada finds that 5 per cent of the total air consumed in drilling and sharpening is taken by the sharpeners. In the Pacific Northwest, at a large lead-silver mine, meter readings were specially taken to determine the percentage, which proved to be a little over 6. Total free air consumption for 90 drills, including that used in blowing for ventilation, for one 8-hr. shift was 1,599,000 cu.ft. Of this total, three No. 8 Waugh sharpeners, running one 8-hr. shift, took 97,200 cu.ft. Each sharpener, in 5 hr. actual running time, sharpened approximately 800 steels. A zinc mining company, in an Eastern state, declares that in one of its mines, during the year 1931, 8.53 per cent of total air "charged to drilling" was consumed in sharpening. For the first seven months of 1932, this figure dropped to 6.78 per cent owing to the reduction in the amount of steel sharpened. Still another operator, mining copper in an Eastern state, finds that the drill sharpeners take 12 per cent of the total air, the rock drills 59 per cent, pumping 1 per cent, air jacks and dumping 2 per cent, and ventilation, blowing, and line losses 26 per cent. This distribution was computed from the number of units operating, length of time of operation, the consumption per unit being actually measured. The figure thus obtained was checked against the total consumption as determined by a recording flowmeter.

The amount of steel consumed per bit sent back into the mine from the sharpening shop should properly be calculated by dividing the weight of a given lot of new steel by the number of sharpened drills turned out from the lot by the time it is used up. Thus all breakage and waste would be included, as should be done. The weight of steel used per bit in driving the Moffat tunnel is said to have been 0.235 lb. This weight will vary with the size of steel used, but the average on any job should be fairly uniform. Some operators possibly underestimate the amount of steel required per bit because they ignore the steel wasted, lost, and scrapped because of fatigue. One manufacturer of detachable bits claims to have gathered data from a number of mining operations and contracting jobs indicating that an allowance of 0.25 lb. of steel per ordinary drill steel sharpened is in general quite fair. He estimates, moreover, that to produce one of his detachable bits takes, on the average, the same amount of steel. Some other detachable bits would take more steel than this one because of their respective designs.

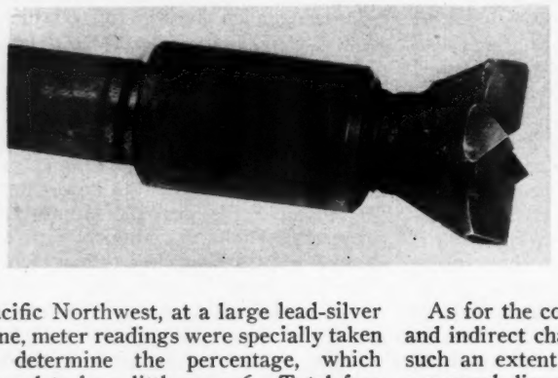

Fig. 3 — Thurston bit, assembled with separate collar and shank. Both bit and shank are partly unscrewed from the collar to show the left-hand thread.

As for the cost of distribution of steel and indirect charges, this will depend to such an extent on local conditions that a general discussion would be pointless. In this connection, the caution displayed by the Anaconda Copper Mining Company in determining the respective costs of drilling with ordinary steel and with Hawkesworth detachable bits is worthy of note. The details of procedure, together with the factors selected for comparing equipment, operating, maintenance, and replacement costs, and the prolonged nature of the tests, which when completed, left no room for doubt as to the economy possible with the latter bit under the conditions prevailing in the mines, are fully set forth in T.P. 274, by Berrien, previously referred to, and will furnish a useful guide for anyone undertaking a similar investigation. The most difficult part of that study is said to have been the determination of the cost of distributing steel. Time studies to this end called for weeks of observation.

The statement of W. J. Coulter (*E.&M.J.*, Dec. 4, 1926) respecting the Granby bit is worth noting. He says, "We have found that it costs twice the value of one bit to take a mine steel through the cycle of sharpening and distributing. Thus a comfortable margin is left." He says further: "An appreciable saving has been made in steel consumption through eliminating loss in the muck, but the most important saving comes in the nipping of steel, which in the past has been expensive because of distant and widely scattered workings."

Mention has previously been made of possible prejudice on the part of the operator against the detachable bit, arising from the fact that the mine is already equipped with steel and shop equipment for sharpening it and that a change may seem expensive. One may say, aside from any question of the possible economy and efficiency of the idea of detachability, that to switch from using ordinary steel to detachable bits should not, at the worst, be any more upsetting than many other changes that have been made in mining and metallurgical practice for the sake of obtaining greater efficiency and lower operating costs. Moreover, the steel on hand can readily be shanked to make it usable with any bit that has been devised. At the worst, some of the sharpening and tempering equipment may have to be scrapped, or held idle to be consumed eventually as replacement parts, as in a shop equipped with two sharpeners, only one of which can be used if the change is made, and that for reshanking.

The story will be different, and the difficulty of changing somewhat greater, if attempt is made to manufacture detachable bits at the mine. So far as known, the Hawkesworth and Granby bits are the only ones being so made. For none of the other bits has manufacture at the mine been proposed. The Granby company has made its bit solely for use in its own workings. The Hawkesworth bit, aside from test work, has thus far been made only by Anaconda for its mines at Butte, where it is regularly used more than once, being reground and heat-treated for the purpose. In contrast, some of the other bits now being made are intended to be used but once and then scrapped, this reducing the shop equipment required to a minimum.

No good reason can be seen today why the adoption of a detachable bit should cause any marked change in such elements of drilling practice as type of bit, bit gage, and size and section of rods. The cross bit has proved most suitable for general standard drilling purposes, and, reflecting this situation, companies and individuals interested in detachable bits have preferred the cross type for their product. Of the most popular shapes of bits—namely, the cruciform or cross bit, the rose bit, the X bit, the double-arc or double-C bit,

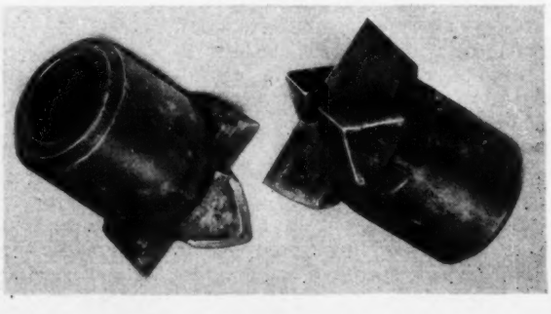

Fig. 4—Two views of the so-called "Erbit." Note how the collar has been crimped on to the bit proper.

the McLellan Kerr Lake bit, and the bull, Carr, and Z bits—only three, the cruciform, double-arc and chisel, are now being made in the detachable form in the United States and Canada. Of these three the cruciform is by far the commonest. The other types can doubtless be made easily enough, if demanded. One manufacturer now making only a cross-type bit has on his program for the future the development of a rose or six-pointed bit and a modified chisel bit. The Granby bit was formerly made in a chisel type, since discarded. As for gage, earlier models of detachable bits ran to larger sizes than bits used in standard drilling practice. This difference is largely disappearing.

The foregoing paragraphs have been written in the hope that a knowledge of the facts will insure a more open attitude of mind toward the detachable bit on the part of those operators who have tacitly assumed that, because this bit did not long ago spring into widespread favor, something must be hopelessly wrong with it, save possibly under exceptional conditions. Evidently, much has been wrong with the bits tried and discarded in the past. The performance of the Hawkesworth bit at Butte, and the more recent progress with other bits elsewhere, can be accepted as fairly convincing evidence that the detachable bit will yet prove a more widely serviceable tool to the mining and quarrying industries than it has been to date. Generalization respecting its possibilities, however, must not be carried too far. Some of the designs developed may prove better suited to certain drilling conditions than others.

Any bit that may be tried is entitled to an exhaustive test before judgment is rendered. When the question arises in any mine office as to whether detachable bits should be substituted for standard steel, knowledge of relative costs of drilling with both methods will permit an intelligent decision. A correct decision can hardly be made unless these costs are known. With accurate knowledge as to what the use of standard sharpened steel means on the cost sheet, the price at which a detachable bit of any given type and make will be worth a trial can be quickly determined.

Models of detachable bits now available to the mining industry include the following: the Hawkesworth bit, controlled outside of the Anaconda company's mines by the Hawkesworth Drill Company, Butte, Mont.; the Thurston bit, made by the Detachable Bit Corporation of America, New York; the so-called "Erbit," developed and sold by the Economy Rock Bit Corporation, Lancaster, Pa.; the Timken rock bit, developed and made by the Timken Roller Bearing Company, Canton, Ohio; the Crusca bit, developed and made by the Crucible Steel Company of America, New York; the "Rip-Bit," developed by George C. Riley, of Montreal, Canada; and the DeWitt bit, made by the DeWitt Detachable Bit Company, Oakland, Calif. The Granby company's bit, previously referred to, has not been used outside of the mines at Anyox, nor has an effort been made to sell it or license its manufacture.

A bit not yet on a production basis and of which little in fact is known is one understood to be in process of development by the Oliver Iron & Steel Company, Pittsburgh, Pa. This company is said to have become interested in detachable bits through making forgings for the Detachable Bit Corporation of America. In similar fashion, the Brown-Lipe-Chapin Company (a General Motors subsidiary), Syracuse, N. Y., which made the "Erbit" for the Economy Rock Bit Corporation, is said to have become interested in bits of this sort and to have embarked on the development of a certain model of its own, since abandoned.

Regarding the statement made in a previous paragraph to the effect that the Hawkesworth bit is available, a word of explanation is necessary. Owing to prevailing economic conditions the Hawkesworth company has been unable to perfect its plans to the end that other companies, aside from the Anaconda company, can be serviced with its bit. Early in 1931, an option agreement was entered into with a group of iron and steel men, and tests, said to be satisfactory, were conducted at certain mines in the Michigan Iron Country. Owing to economic conditions, however, the option was not consummated. The bit is here classed as available to the mining industry because of the extensive use made of it by the Anaconda company, which owns all rights pertaining to its use within the limits of its mines, and which, it might be added, has discarded all of its formerly used standard drill steel. A bit and shank is shown in Fig. 1.

Very little change has been made in the principle of design of the Hawkesworth bit since the presentation of Mr. Berrien's paper, previously mentioned, at the February, 1930, meeting of the American Institute of Mining and Metallurgical Engineers. Briefly, the bit, which is double-taper cruciform in type, consists of a single machined forging, which is attached to the shank by what has been described as a dovetail joint modified so that it cannot become loose, the bit being the tenon and the shank the mortise. A third member or piece is not required. According to Berrien, the bit was formed in a screw press from a 1½-in. light-section cruciform steel, trimmed, annealed to maximum machinability, cleaned of scale, milled and, finally, ground automatically. Since Technical Paper No. 274 appeared, however, the screw press for forging bits has been superseded by an Ajax automatic forging machine and 1¼-in. light-section cruciform steel has been substituted for the 1½-in. steel. The 1¼-in. steel is fed, after being heated, into the forging machine, which cuts off the proper amount of stock, forges the bit, and delivers it to a conveyor, which carries it to the flash trimmer, thus doing away with the costly operation of cold-shearing the bars into slugs preliminary to pressing. The change from 1½-in. to 1¼-in. steel has resulted in the formation of less flash and therefore an increase in the number of bits to be obtained from a pound of metal.

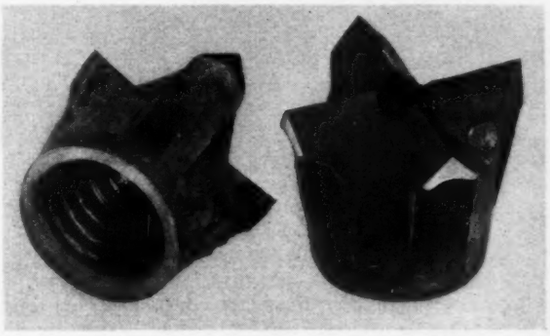

Fig. 5—The Timken rock bit.

The Hawkesworth bit is intended to be resharpened by grinding when dull, this practice serving to provide bits of the smaller sizes needed to permit the required drops in gage. Starters are made in 1⅞-in. gage, the finishers being approximately 1⅜-in. The bit size limits are not held closely, because of excess grinding required, and may vary slightly at the Anaconda company's different mines. The steel used for the bits runs 0.8-0.9 per cent carbon and 0.2-0.3 per cent manganese. For the rods, a 1-in. quarter-octagon steel, running 0.75-0.85 per cent carbon and 0.25-0.35 per cent manganese, was selected by Anaconda as standard for all stopes, drifts, and raises, except for sinking and plugging. Further than this, detailed description of the bit is unnecessary.

The remaining bits are markedly different from the Hawkesworth in design. The Granby bit, shown in Fig. 2, is in a class by itself as compared with the others. It is being used today in the Granby company's Hidden Creek mine for drilling 20-ft. holes for big blasts. In this application, it is considered economical, inasmuch as the

spacing of holes can be made proportionate to the bigger volume of the hole and the work of nipping is reduced. It has been found uneconomical for development and general mining work, and for drilling hard rock effectively.

This bit is of the cross type and is simple in design. It is backed by a tapered shank which fits into a correspondingly tapered socket in the end of the rod, without being pinned or locked. On one side of the shank of the bit, and in line with the valley between the wings, a water groove is pinched by forming tools. The bit is resharpened. Most of the bits, when originally made, are turned out in starter size ($2\frac{3}{4}$ in.) and through use and resharpening gradually wear themselves down to finishers (2 in.) after an average life of fifteen to twenty sharpenings. No definite sequence of gages is provided. It might be added, to avoid possible confusion, that the Granby bit was developed from a straight chisel-type bit, having a high center, which was known as the "Numa" and which was discarded.

The extent to which the Granby bit is used at Anyox is shown by the following figures: In 1930 the sharpening shop sharpened 27,060 regular mine steel and 10,600 detachable bits per month. In 1931, the corresponding figures were 24,150 and 4,150 respectively, and, in the first six months of 1932, 21,200 mine steel and 9,810 detachable bits per month, respectively.

The Thurston bit, shown in Fig. 3, has a development history stretching back over several years, to go into which is quite unnecessary. Its sponsor has been the Detachable Bit Corporation of America. The bit requires a separate coupling or collar for attaching it to the shank. Bit and shank have each a left-hand male thread, which engages with a corresponding female thread in the coupling. A shoulder on the inner end of the bit abuts against the end of the shank, when the unit is assembled, so that the special round threads are not subjected to punishment, this feature also facilitating removal from the shank. The bit, as manufactured in the past, is said to be of a special abrasion-resisting, high-temper alloy steel; made in three types—namely, cruciform, double-arc and chisel. It may be sharpened two to four times, as a good many users are reported to be doing. While the company has not emphasized this feature, it believes that it will become general practice. Bits and couplings are thrown away after they are used up. The company has maintained at its shop a service for rethreading and reshanking the rods. It intends to make arrangements in different parts of the country to take care of such work in the future. The coupling is of an alloy steel, reported to be harder than the steel of the rod, which is of a special fatigue-resisting quality. The bit has been furnished in gages ranging from $1\frac{1}{2}$ to 3 in., inclusive.

According to report, the Thurston bit has been satisfactory in general as to service. Manufacture is said to have attained a rate as high as 4,000 bits per day. Up to a few months ago, according to an employee of the company, a few million bits had been turned out, including those used for trial and testing during the development period.

One should logically consider the Economy company's Erbit next. In the Erbit, Fig. 4, the bit and collar are attached together by hot-crimping the collar on the bit, as is well shown in the photographic reproductions. According to Walter C. Rea, who organized the Economy corporation to develop this bit, a distinguishing feature is that the collar section is softer, or of lower Brinell nature, than the threads on the drill rod. This was developed to eliminate frequent rethreading of the rod, inasmuch as the softer member takes the wear. Mr. Rea claims to have had rods that have been used with 100 or more bits without showing appreciable wear. The steel used in the bit is an alloy said to be selected to give maximum hardness and abrasion resistance, together with a tough core. The rods supplied by the company are of a relatively lower carbon steel. Users can employ whatever rods they choose and thread them in the field with a tool specially devised for the purpose.

The Erbit is made only in the cruciform type, and in five gages for $\frac{7}{8} \times 3\frac{1}{4}$-in. jackhammers and six gages for both 1 x $4\frac{1}{4}$-in. jackhammers and the lugged-type drifter. Thus far it has been sold in limited quantities.

The Timken rock bit is illustrated in Fig. 5. The idea of the design was conceived within the Timken organization, where it was under development for over three years. During that period, more than 30 different bits are said to have been designed, manufactured, and tested before the adoption of the present bit. A considerable number of steel analyses were also tried prior to selection. The steel chosen is one that is difficult to heat-treat without special equipment. The bit is of the cross type and is screwed by means of a female thread directly on to the male-threaded shank or rod, without use of a third part. Its thread is left-handed and is said to be designed for strength and easy detachability. The end of the shank is shouldered, and against it bears the base of the bit so that the blow is not transmitted through the threads. This design is said to provide maximum air clearance and to have the advantages that the bit is rigid on the rod and the impact of drilling is not concentrated on the center or "weakest part" of the bit, which, it is held, might cause cracking. A result claimed is longer life of thread.

According to the claims of the manufacturer, the design of the Timken bit permits gages as small as $1\frac{1}{4}$ in. to be furnished. The bit is made in gages of $\frac{1}{8}$-in. intervals, including eleven sizes from $1\frac{3}{8}$ to $2\frac{5}{8}$ in. with a 1-in. thread, for use with $\frac{7}{8}$-in. or 1-in. hexagon steel and 1-in. or $1\frac{1}{8}$-in. round steel. A $1\frac{1}{4}$-in. gage bit with $\frac{7}{8}$-in. thread is for use only with short lengths of $\frac{7}{8}$-in. hexagon steel. Gages ranging from $1\frac{1}{2}$ in. to 3 in. will, according to present plans, also be developed for $1\frac{1}{4}$-in. steel.

The maker of this bit undertakes to supply $\frac{7}{8}$-in. steel and 1-in. hollow hexagon steel and 1-in. and $1\frac{1}{8}$-in. hollow round steel in any length, for use with the bit; or will convert steel already in the bit user's possession; or will furnish instructions and tools to enable the user to convert his own steel. Such steel as the user might have on hand would have to be of proper quality, free from defects and properly heat-treated, if it is to stand up on conversion. The maker holds that when rod threads must be re-conditioned, the proper way is to cut the thread off entirely and normalize the steel to remove strains before upsetting, threading, and heat-treating. This bit has been on a production basis since early last spring. Its manufacture is not to be licensed.

Fig. 6 illustrates the DeWitt bit. It is the only bit now on the market, so far as known, using a bayonet lock for attachment to the shank or rod. It was developed by Charles W. DeWitt in Mexico four years ago. Since then, the original idea has been changed but slightly.

A simple locking device is the distinguishing principle of design. It comprises, as to the bit itself, two wide, tapering, locking wings set diametrically opposite each other inside the bit, which is thus the female end. On the end of the shanked steel are similar wings. The bit is placed in the shank and locked by means of a quarter turn. Rotation keeps it locked. To release it, when dull, a few blows with a hammer and a quarter turn in the opposite direc-

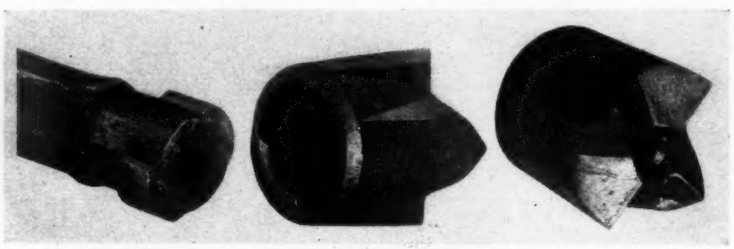

Fig. 6—The DeWitt bayonet-lock detachable bit and shank.

tion allow the wings to slip out. The only changes made in design have been in proportioning the walls and wings.

This bit is made in six gages and in the cross type only, although the single-edge or Carr bit can also be supplied. Gage ranges from $2\frac{1}{4}$ down to $1\frac{5}{8}$ in., in $\frac{1}{8}$-in. intervals. Bits can be resharpened three times on an average, this depending on the extent of first use. Sharpening can be done on an ordinary blacksmith emery wheel. If this be well done, retempering is said to be unnecessary. A high-carbon tool steel is used in the bit.

Bits are furnished to users direct. Manufacture will be licensed, if a company prefers to install its own plant. Shanked steel ready for use with the bit is supplied if wanted. Dies for shanking the steel can be obtained by companies having their own drill sharpeners. Any size of hollow steel may be used—hexagon, round, or quarter octagon.

An interesting bit is the Riley "Rip-Bit," development and testing of which have recently been completed and which is now on a production basis. Mr. Riley, who is the managing director of Holman Machines, Ltd., of Montreal, has long worked on the problem of its development. As designed, a stud is used to attach the bit to the shank, as shown in Fig. 7. The bit is of the cruciform type, this form having been selected as best for general drilling, and is intended to be scrapped with the stud after it has been used but once. Production cost, according to report, will be low enough to permit this.

Bit and stud are assembled by the makers and shipped as one unit. The stud is locked into the bit, so that it is impossible for a miner to arrive at a working place and find he has a sufficient supply of bits but is lacking the full complement of studs. An additional advantage claimed is that only two parts have to be handled—namely, the bit unit and the drill rod, which remains underground with the drill for many shifts. Moreover, there is nothing to be misplaced or to be lost underground. As will be seen in Fig. 7, the bit abuts directly against the shank, so that the blow is not passed through the stud.

The bit is characterized by a simplicity, which nevertheless is said to belie the costly process of evolution by which it was attained. Despite the appearance of the stud, Mr. Riley says, repeated tests have shown that it will stand up without breakage for at least 30 min. In these tests he claims to have used the heaviest drills made in the United States or abroad, 100 lb. air, and the hardest rock he has been able to find in Canada, England, and the Continent. Bit and stud are of extra clean carbon steel, properly heat-treated. Many different alloy steels were made into bits and tested over a considerable period. Though many of them possessed valuable heat-treatment characteristics, affording ease in handling through wide temperature variation without damage, none of them, according to Mr. Riley, stood up against abrasion any better than the best carbon bits. As resistance to abrasion is the most important factor in a successful bit, Mr. Riley says he was forced to adopt the steel now used.

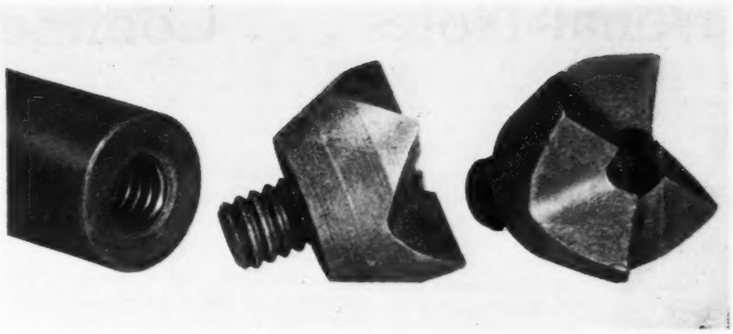

Fig. 7—Two views of the Riley "Rip-Bit" and the shank end of one of the rods

The finished bit and stud are supplied in all gages direct to users. Mr. Riley considers it economically impossible to make this product at a mine, saying that he does not propose to make a steel manufacturer out of a miner. The skill required in making and heat-treating bits and studs is a prime requisite, calling for centralized manufacture. Rods also will be furnished in a steel made primarily to resist fatigue. Also an inexpensive threading device the use of which calls for only elementary skill.

The question of gage Mr. Riley considers important. He holds strongly to the opinion that nothing less than $1\frac{1}{4}$-in. powder should be used in underground work. Though prepared to make as many gages as called for, he thinks it practicable as a rule to start in hard, abrasive rock with a $1\frac{7}{8}$-in. and bottom with a $1\frac{3}{8}$-in. bit, by $\frac{1}{8}$-in. changes; and in softer rock, to start with $1\frac{5}{8}$-in. and finish, say a 9-ft. hole, with two additional $\frac{1}{8}$-in. drops in gage.

Steel for the rods, which have been named "Seldom-Hoist," will be furnished in a quality having maximum resistance to fatigue, and in any length and section. Experimental work has afforded much useful information as to the energy-transmitting properties of various sections and diameters; also as to the relative resistance to breakage of different sections and diameters. The investigation has not as yet been concluded. In discussing these matters, Mr. Riley said it is not generally recognized that drill upkeep charges are largely influenced by the surface area of the steel on which the blow of the piston or the anvil block is received. Within reason, the larger this area, the lower the costs.

Almost as this article was going to press, word was received that the Crucible Steel Company of America had just completed the development of a detachable bit of its own, which it had named the "Crusca." This bit is shown in Fig. 8. The company has long been engaged in making rock-drilling steels, and, through its acquaintance and contact with the mining, quarrying and construction industries, has had an opportunity to consider their needs. Its president, F. B. Hufnagel, has vouchsafed the information that the company officials have closely followed the development and introduction of various detachable bits and, finally, with the aid of their metallurgical and operating staffs, have undertaken to offer the bit named, together with forged collar shanks in lengths from 2 to 24 ft., to users whose particular kind of work may require this kind of drilling equipment. Satisfactory performance is said to have been obtained in laboratory and field tests.

Only one bit remains to be discussed, of those mentioned at the outset, namely, the bit stated to be under development by the Oliver Iron & Steel Company. Report has it that the bit proper consists of two pieces of steel, of different analyses, forged securely together and provided with a male thread, the rod carrying the female thread on the upset end. Apparently it has not yet emerged from the embryo stage.

Fig. 8—The "Crusca" bit and shank

Personal Notes . . . Comments . . . Criticism

Dr. Frederick M. Becket, president of Union Carbide & Carbon Research Laboratories, has been nominated for president of the A.I.M.E. for 1933. Dr. Becket was born in Montreal in 1875, and is a graduate of McGill, subsequently taking his master's degree at Columbia. The latter university bestowed its D.Sc. upon him in 1929, in recognition of his contributions in the field of electrometallurgical development of the ferro-alloys. Dr. Becket was president of the American Electrochemical Society in 1925-26 and Perkin medallist in 1924.

H. T. Dickinson, of Johannesburg, consulting engineer for DeBeers Consolidated, was recently in New York.

H. S. Munroe, consulting engineer for Rhokana Corporation, has returned to Northern Rhodesia, to act as *locum tenens* for the general manager, who is on vacation.

Dr. R. Lockhart Jack and **J. Hockey,** of the staff of Broken Hill, Pty., are engaged making geological surveys of the Enterprise, Hannan's North, and King of Creation Mines in Kalgoorlie, W. A.

Harry Bible has resigned as mechanical superintendent for Miami Copper. **J. Luchessa,** formerly electrical superintendent, has been placed in charge of both the mechanical and electrical departments.

C. W. Van Law made a tour in eastern Canada during the first week of November with his old friend and associate of Peruvian days, **E. J. Carlyle,** secretary of the Canadian Institute of Mining & Metallurgy. At the invitation of the Institute, he addressed the branches at Montreal, Toronto, Porcupine and Kirkland Lake, his subject being gold mining in New Guinea, from which country he returned recently. The meeting at Kirkland Lake signalized the formation there of a new branch of the Institute.

Donald B. Gillies, president of the Corrigan, McKinney Steel Company, Cleveland, Ohio, has been made president of the Newton Steel Company, a subsidiary of the Corrigan, McKinney company. He also retains the presidency of the N. & G. Taylor Company, also a subsidiary. The affiliation of Corrigan, McKinney with the Newton company assures the latter company a permanent source of supply, and Corrigan, McKinney acquires another outlet for its steel. The Newton company has sheet mills at Monroe, Mich., and Newton Falls, Ohio. The N. & G. Taylor Company manufactures tin plate at Cumberland, Md.

E. W. Todd has resigned as assistant manager at the Lake Shore mine, Kirkland Lake, Ont.

R. A. Archbold, general manager of New Guinea Goldfields, is on a business visit to Great Britain.

C. W. Van Law, consulting engineer, has established headquarters at 29 Broadway, New York City.

Alan Bateman, professor of economic geology at Yale University, has concluded an examination of the mining properties of American Metal in Mexico.

Henry Briggs, who contributes to this issue an article on fire-fighting equipment, is Hood professor of mining at the University of Edinburgh and professor of mining at the Heriot-Watt College. In addition to a distinguished academic career, Professor Briggs has acted as consultant to the British Government, and as research director in mine-rescue work from 1917 to 1920; and as a member of the oxygen research commission, 1918-21. He was one of the technical advisers to the War Office during the War, and is the inventor of safety and survey apparatus for use in mines. After graduation from the Royal School of Mines with honors, he later received a doctorate in science at the University of Birmingham and a doctorate in philosophy at the University of Edinburgh. He is a fellow of the Royal Society of England.

A. R. Williams has been appointed manager of Neangar Gold Mining, Bendigo, Vic., Australia.

Louis Notte is directing the installation of rock-crushing and ore-treatment equipment and the introduction of new mining methods at the properties of Carrieres de l'Ouest, in France.

P. M. Arthur, American Zinc Company of Tennessee, was recently elected vice-chairman in charge of engineering of the executive committee of the mining section of the National Safety Council. Other appointments included: **Frank B. Dunbar,** Pickands, Mather & Company, vice-chairman; **J. W. Alt,** Calumet & Hecla Copper, vice-chairman, and the following as members at large: **W. H. Comins,** National Lead; **M. F. Fairlie,** Mining Corporation of Canada; **H. C. Henrie,** Phelps Dodge Corporation; **George Martinson,** Pickands, Mather & Company; **D. D. Moffatt,** Utah Copper, and **Howard I. Young,** American Zinc, Lead & Smelting.

Dr. R. W. Brock, Dean of the Faculty of Applied Science at the University of British Columbia, has been engaged by the British War Office to complete a geological survey of Hongkong, which he initiated in 1923, and which has since been intermittently carried on by other members of the geological staff of the University of British Columbia. During Doctor Brock's absence, Prof. J. M. Turnbull, head of the department of mining, will serve as acting dean of the faculty, and Doctor Brock's duties as lecturer in economic geology will be undertaken by Dr. Victor Dolmage, formerly in charge of the British Columbia branch of the Geological Survey of Canada. **Dr. T. C. Phemister,** professor of mineralogy, has also been granted leave of absence. His place will be filled by **Dr. Harry V. Warren,** who, after graduating from the university in 1926, proceeded to Oxford as a Rhodes scholar, where he obtained his doctorate and subsequently was awarded the Commonwealth Traveling Fellowship at the California Institute of Technology.

Huntington Adams is leaving New York for Menton, France, soon.

Harry A. Clark is at the Calumet Hospital, Douglas, Ariz., recovering from a severe illness.

L. V. Waterhouse, a director of Bululo Gold Dredging, has joined the directorate of General Airways, Ltd.

T. F. Donnelly, general manager for Cia. Minera Kildun Y Anexas, Matehuala, S.L.P., Mexico, is in New York.

Michael H. Loveman, of Joplin, Mo., was recently at Atlanta, Idaho, examining the Monarch group of mines there.

D. C. Jackling has been awarded the John Fritz Gold Medal for 1933, for notable achievement in the mass production of copper from low-grade ores. This medal, the highest honor in the American engineering profession, is awarded by a board composed of representatives of the four national engineering societies. Former recipients include President Hoover, Thomas A. Edison, Guglielmo Marconi, Michael I. Pupin, Lord Kelvin, Orville Wright, Elmer A. Sperry, John F. Stevens, and John J. Carty. Mr. Jackling in 1919 received from Congress the Distinguished Service Medal for his work during the War as Director of Explosives. In 1926, the Mining & Metallurgical Society of America awarded to him its gold medal for distinguished service in the development of low-grade mines. In 1930, he was named the Saunders Medallist, for "achievement in mining."

James Orr Elton, general manager of International Smelting, Salt Lake City, has been nominated to receive the James Douglas Medal, awarded by the A.I.M.E., "for conspicuous work in the treatment of smelter fume and in the production of electrolytic zinc; for the development of successful methods for preferential flotation of lead-zinc-silver ores and oxidized lead-silver ores; and for improvement in lead smelting."

Harold Elford, who contributes the article in this issue on "Osmiridium in Tasmania," is a graduate in metallurgy of the University of Adelaide, South Australia. After three years with Electrolytic Zinc, as research engineer and mining assistant to the general manager, he joined the staff of the Commonwealth Government Development & Migration Commission, specializing in the development of the mining industry.

Bewick, Moreing & Company, general managers for Zinc Corporation, draw attention to an error in a news note in the August issue. Owing to conditions in the world's market for non-ferrous metals, the Broken Hill mine of Zinc Corporation worked only 75 per cent of normal time, but operations were almost continuous throughout 1931.

John Becker, recently assistant underground manager at Roan Antelope, has been appointed underground superintendent of the N'Kana mine of the Rhokana Corporation. Mr. Becker has been visiting London, and is now returning to Africa to take up his new duties.

John Knox, formerly assistant general manager, has been appointed general manager of the operations of Hollinger Consolidated. **A. F. Brigham,** formerly general manager, has been retained by the company as consulting engineer.

Gomer P. Jones, manager for over 30 years of the Nickel Plate mine at Hedley, B. C., has retired. Mr. Jones recently started from Vancouver on a world tour via Australia, in which country he was born.

W. Baragwanath, previously Director of the Geological Survey, has been appointed Secretary of Mines, Victoria, Australia.

Walter H. Aldridge, president of Texas Gulf Sulphur, has been named as the Saunders Medallist for 1933 by the directors of the A.I.M.E. The citation, as drawn by Louis S. Cates, chairman of the Medal Committee, is "for trained judgment in recognizing the values of undeveloped mineral resources; for skill in developing and operating mines; for eminent success in organizing and directing outstanding mining enterprises, involving difficult engineering and economic problems."

Leaching Fine Sand

The Editor:

One reading your editorial in the issue of October, 1932, entitled "More Light on Wet Extraction Processes," would receive the impression that the feasibility and advantages of percolation for the leaching of finely ground sand has been only recently realized by metallurgists. In a paper by Charles Butters and E. M. Hamilton before the Institution of Mining and Metallurgy in 1904, "On the Cyaniding of Ore at El Oro, Mexico," you will read: "It is interesting to note that the sandy part of the product which was mostly finer than 200 mesh did not show the smallest difficulty in leaching (percolation), using a charge 5½ ft. deep." And again: "It has been shown that sand 70.7 per cent of which will pass a No. 200 mesh screen will leach freely in charges 5½ to 6 ft. deep provided that all amorphous or clayey material is separated from it. It is only a question of a thorough separation."

As regards extraction, in the same paper: "It is remarkable that in these tests a better extraction is obtained on the silver by leaching (percolation) than by agitation."

In an article I wrote to *Mining & Scientific Press* (Aug. 21, 1907), I said that "The reason why the leaching (percolation) treatment has been apparently unsuccessful in some cases where fine grinding is practiced is probably because a good separation of sand from slime has not been obtained." In Hamilton's "Manual of Cyanidation" (1920), attention is again called to the fact that "a good proportion of the sand minus 200 mesh is perfectly leachable (percolable) if properly freed from slime."

Thus from 1904 onward the general principles to which you now call attention have been recognized, and are on record in public articles and transactions. No doubt other metallurgists could bring similar but independent testimony to the same effect.

E. M. Hamilton.

San Francisco, Calif.

Regrets are tendered to Mr. Hamilton that the editorial in question could be interpreted as announcing a discovery, and thanks are due to him for assembling the references in the literature of a phase of processing technique in which he has been a distinguished contributor.

Justification for drawing attention to Homestake results may be found in the specific nature of the information used as the text for the editorial comments—an extraction by gravity leaching of 90 per cent of the gold in 260,000 tons of minus 200-mesh sand per annum; and the extension of research into the finer ranges of classification culminating in the knowledge that no less than 85 per cent of minus 260-mesh plus 325-mesh product is a sand, offering no obstacle to even percolation and successful extraction.—Editor.

H. C. Enos has returned to Chatsworth, Calif., on the completion of professional work at Yuma, Ariz.

W. E. Flanagan, formerly on the mill staff of New Jersey Zinc, is now plant engineer for the Walter Baker Chocolate Company, Dorchester, Mass.

Robert M. Peterson, who contributes an article in this issue on mining practice at the Roan Antelope mines, in Northern Rhodesia, is a graduate of the Michigan College of Mines. Prior to War service he was on the staff of Calumet & Hecla, as efficiency engineer. In 1919 he was appointed superintendent of mines for Witherbee, Sherman & Company, Mineville, N. Y., remaining in that capacity until the suspension of operations in 1922. His next post was that of mine superintendent for the Chateaugay Iron Company. In 1928 he was appointed mine superintendent at Roan Antelope.

Herman A. Wagner has been awarded the Clausen Medal by the American Association of Engineers, for service in the advancement of the social and economic welfare of engineers.

G. St. G. Sproule is now associate professor in the department of metallurgy, McGill University, Montreal.

Alaska Juneau

The Editor:

In your issue of September, 1932, is an article on the Alaska Juneau by P. R. Bradley, on "History, Organization, and Outlook." In it he makes some misstatements of fact regarding which I think, in simple justice, you should present the following statement:

1. It is not true that the "plans of the Alaska Gold Mines Company began to wean Mr. Bradley's partners away from the mentioned and agreed upon program for Alaska Juneau financing and mill construction."

2. It is not true that the disagreement over financing impaired Mr. Bradley's health.

3. It is not true that funds had been provided by public subscription against Mr. Bradley's wishes.

This disagreement was duly set forth in the columns of *Mining & Scientific Press* by Mr. Bradley under date of Feb. 9, 1918. An answer to his statement was also published in *Mining & Scientific Press* on March 9, 1918, over the signature of J. H. Mackenzie. Mr. Mackenzie's statement has never been refuted. From it I excerpt the following:

"For several months prior to the flotation, the Alaska Juneau was borrowing money and becoming further involved in debt. At the time of financing it owed $600,000 and everybody connected with the company was concerned, including Mr. Bradley, as to how we were going to carry on the work and pay off the loans. At a conference held on Jan. 23, 1915, William H. Crocker, Wellington Gregg, Jr., F. W. Bradley, M. L. Requa, and J. H. Mackenzie being present, the financial condition was discussed. Mr. Bradley had just returned from New York, where he had refused an offer made by B. M. Baruch, of a $3,000,000 bond issue, convertible into shares of $8.75—the bonds not to draw interest before July 16, 1916. Mr. Crocker, the largest shareholder in the Alaska Juneau Gold Mining Company at that time, asked on what grounds he had refused this offer. Bradley replied that Ogden Mills and he wished to retain their present percentage in the property and feared Mr. Baruch intended to get control of the shares and turn the man-

agement of the property over to the Alaska Gold Mines Company. Mr. Crocker replied that it was very dangerous for an enterprise to contract large call-loans and that it was almost an axiom that any enterprise in which this practice was pursued sooner or later would be sure to get into trouble. The property was looking exceptionally well at this period and the feed to the 50-stamp pilot mill was averaging over $1.50 per ton; in fact, never before nor since has the Alaska Juneau looked as well as it did for the twelve months between May, 1914, and June, 1915, and any opposition to public financing that Mr. Bradley may have made was because he thought, as we all did, the property would come up to our expectations and was a good permanent investment, and of the further fact that he feared he might lose control."

It is true that Mr. Bradley was ill and did make a trip to Honolulu.

I again excerpt from Mr. Mackenzie's reply:

"On April 6, 1915 (more than two months prior to Mr. Bradley's departure for Honolulu), he signed a contract on behalf of the Alaska Juneau Gold Mining Company, with George O. Bradley, representing Bradley, Bruff & Labarthe, wherein it was mutually agreed that the latter was to 'undertake the design and superintend the erection of the 8,000 tons daily capacity milling plant for the above mining company, together with the necessary auxiliary departments, such as power-plant, crushing plant, etc.' In a letter to Mr. Requa, on April 6, 1915, Mr. Bradley said:

"'You will be pleased to learn that we have today closed a contract with George Bradley, and his associates, for the building of our new mill in accordance with enclosed copy of contract itself, and in accordance with enclosed copy of my letter of today to Phil.'

"On the same day he wrote his brother, Philip R. Bradley:

"'Mr. Mackenzie and myself have made a definite business arrangement with George Bradley to take complete charge of the designing and erection of our new milling plant, to have a capacity of 8,000 tons per day. This means that George Bradley, or his representatives, will have charge of all experimental work in the present mill, and of all work of every kind in connection with the new rock-crushing and milling plant. You are to give George Bradley, or his representatives, every assistance and every facility that you see they may need or that they may have occasion to ask for.'"

I do not desire, at this time, to go further into the details of this controversy. I am simply trying to point out, in justice to J. H. Mackenzie, who is now a hopeless invalid and confined to his house, that Mr. Bradley was quite as much responsible for the policy of the milling operations as was any one else.

It is unfortunate after all these years that this old disagreement should be raked up again and the same misstatements made in the year 1932 that were made and refuted by Mr. Mackenzie in 1918.

I do not minimize the credit due Mr. Bradley for having since made a success of the enterprise, but I cannot permit the statements of Mr. Philip Bradley to pass unchallenged. The original controversy was one that should never have gotten into public print. Once having been printed it should have been forgotten at the earliest possible moment. Referring to it after all these years and repeating statements that are entirely erroneous is most regrettable. M. L. REQUA.

San Francisco, Calif.

The Editor:

Mr. M. L. Requa is incorrect in assuming that either myself or my brother, F. W. Bradley, wrote the editorial reference or anything else in the *Mining & Scientific Press* as of Feb. 9, 1918, or as of any other date. My brother has never made any printed statement regarding this so-called controversy and this is my first public reference to it.

The first article relating to this so-called controversy appeared over the name of J. H. Mackenzie in *Mining & Scientific Press* of March 9, 1918. This article was refuted by Mill Superintendent V. C. Clauson's article on the "Alaska Juneau Mill" as regards mill failures, due to J. H. Mackenzie, in *Mining & Scientific Press* of March 7, 1921; and the other matters said to be in controversy were again editorially referred to in the same number of *Mining & Scientific Press*, and no public notice of these refutations was ever made by either J. H. Mackenzie or M. L. Requa. P. R. BRADLEY.

San Francisco, Calif.

■

Obituary

Henry Ellsworth Wood

The death in Sacramento, Calif., on Oct. 6, of Henry E. Wood marks the passing of a metallurgist who had gained a reputation for ability in assaying, and in developing machines and methods for treating ores. The record of his activities is almost a history of the development of non-ferrous metallurgy during the last fifty years.

Mr. Wood was born in Joliet, Ill., in 1855, and received his technical training in the Sheffield Scientific School, graduating in 1876. Prior to graduation he spent a vacation in Colorado, accompanying the Powell expedition in 1868, being the youngest member of the party—a trip that led to his location in the West a few years later. In 1878 he opened an assay office in Boulder, Colo., but a short time later moved to Leadville, where he built and equipped a complete and efficient ore-testing and treatment plant. The first Wilfley table was tried out in Wood's office, and the late Arthur Wilfley was one of his staunchest friends. The Hardinge conical mill was developed in Denver with Mr. Wood's assistance; and the Wood plant was the scene of almost all the experimental and development work of the late Dr. Lucien I. Blake in connection with the Blake-Morscher electrostatic separator. Some of the earliest investigations in flotation was also made there, when Mr. Wood imported the first Elmore vacuum-flotation unit to be used in the United States, antedating by several years the introduction of the more familiar frothing process. Mr. Wood himself patented and perfected a process of surface-tension flotation, of which several commercial installations were made.

Mr. Wood's main interest and success was in the working out of process flowsheets. One of the first of these for the Colorado Zinc Company, established in West Denver in the early nineties, involved the problem of treating the relatively low-grade sulphide gold ores then coming into prominence in Colorado, in which zinc was a detriment. The removal and recovery of the zinc as a commercial product was achieved by a combination of gravity, magnetic, and electrostatic methods. Other process work was on American tungsten ores, in which he supplemented gravity methods with high-power magnetic treatment, and his work in collaboration with the late A. B. Frenzel on the concentration of some of the first Gilpin County pitchblende ores that were mined for radium content. A project with which he was associated was the establishment in eastern Canada of a treatment plant that was the major source of supply of the molybdenum used by the British for smoke shells and tracer bullets during the War.

In 1920, following a fire that almost destroyed his testing plant, Mr. Wood limited his activities to the assay office in Denver, still carried on under the name of the Henry E. Wood Assaying Company, and became interested in the oil industry. With his son, Ellsworth Wood, he established refineries at Lewiston, Mont., and Vancouver, B. C. For the past five years Mr. Wood had been active in gold mining in California, where he made his home.

In July of this year he was made an Honorary Senior Member of the A.I.M.E. H.B.B.

Louis Duparc, professor of mineralogy at the University of Geneva; at Geneva on Oct. 20; aged 66.

Ralph Anderson, underground manager of Lake View & Star, Kalgoorlie; at Perth, Western Australia.

Charles Pearce, discoverer of the Cracow goldfield; at Mount Morgan, Queensland, recently; aged 63.

R. G. Eggleston, onetime treasurer and director of Canada Copper Corporation and New Dominion Copper; at New York on Oct. 7; aged 76.

Walter M. Brodie, in charge of the Shepherd mining interests at Batopilas, Chih., Mexico, from 1881 to 1891; at Washington, D. C.; aged 82. Mr, Brodie was born in Holland, of Dutch and Scotch parents, educated at the School of Industry, Amsterdam, and the Clausthal mining school, in Germany.

Henry C. Cutting, first president of the San Francisco Mining Exchange; at Berkeley, Calif., from the results of an automobile accident; aged 62. Mr. Cutting was graduated as a mining engineer from the Univeristy of Nevada in 1891, and is the author of several treatises on financial subjects.

A. Polhemus Cobb, senior vice-president of New Jersey Zinc and former president of American Zinc Institute; at Tarrytown, N. Y., on Oct. 6; aged 68. Mr. Cobb entered the service of New Jersey Zinc when 18 years of age, 50 years ago. He became secretary of the company in 1894 and vice-president in 1912. He was one of the founders of American Zinc Institute, of which he was a director and a member of the executive committee at the time of his death.

INDUSTRIAL PROGRESS

Conveying and Elevating With Single Unit Possible by Redler System—Saves Power

En masse conveying by the Redler system is probably unknown to most mine, mill, and smelter executives and engineers. One who has not seen it actually in use may find difficulty in believing it can accomplish what is

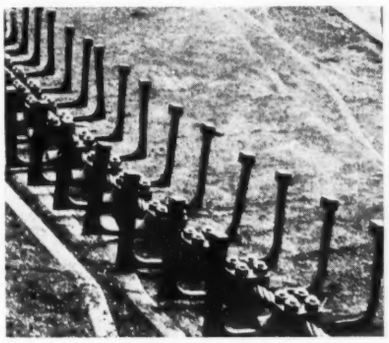

Fig. 1—This Redler conveyor, using U-shaped flights (removed from the trough) has been installed by Rhodesian copper mine

claimed for it, so different is it from ordinary ways of conveying and elevating materials. A demonstration, somewhat limited as to scope, was recently made in New York by the Redler Conveyor Company, subsidiary of the Pneumatic Scale Corporation, of Norfolk Downs, Mass., licensee. Redler Patents, Sharpness, England, control the Redler patents in England. The latter has no connection with the American company. All manufacturing in the United States is done by Stephens-Adamson Manufacturing Company, Aurora, Ill., who are also licensed to sell.

By the Redler system, which is said to have had its beginning in England in flour milling ten years ago, the material conveyed is induced to flow in a compact, quiescent mass through a trough or pipe along any predetermined path—horizontal, vertical or inclined—with a relatively small amount of power. Conveying and elevating can thus be done with the same unit. The materials that can be thus carried vary widely—among them flour, cement, coal, and gypsum. Those actually handled, including the four mentioned, have varied in weight from 4 to 100 lb. per cubic foot; in moisture from a dry state to very wet; and in size of fragment up to as big as 3 in. No trouble has been experienced from abrasion.

The moving impulse is derived from flights made of malleable steel spaced at intervals of several inches along an endless plow-steel wire rope, which is driven by a sprocket. This rope is attached to each flight at either its bottom, its side, or its center, depending on its design. The flights are either U-shaped, H-shaped, or are circular loops. The surprising thing is that flights of such design, occupying so small a per cent of the cross-sectional area of the bed of material moved, should suffice to cause that bed to move continuously and quiescently, not only along a horizontal path but also on an incline and even vertically. The movement is equivalent to flowing because the particles do not move, or eddy, with respect to one another. The ability of the flights to produce and maintain this motion of the material is understood to depend on the arching or side pressure of the material against the sides of the covered trough in which it moves.

Spacing of the flights on the rope is largely determined by the pitch of the sprocket teeth. For practical considerations, it is more or less standardized.

Fig. 1 shows a section of a conveyor having U-shaped flights, styled the Uniflow, which is being installed by a copper mine in Northern Rhodesia. After passing around the driving sprocket, the rope and flights, with the latter upside down, return in the upper part of the trough, as shown in Fig. 2 (where the cover of the trough has been removed), the ends of the inverted flights riding on a renewable wooden strip attached to each side of the trough along a midway line. This type of conveyor is used for conveying either on the horizontal or on inclines not steeper than the angle of repose of the material. The depth of bed will be somewhat greater than the height of the flight arms, save where the conveyor is inclined more steeply than the angle of repose, in which case the depth will not exceed the flight height. Save in the latter case, the depth of the bed of material will be roughly equal to its width. The type of conveyor illustrated has thus far been made in widths up to 23 in. Conveyor speed is 30 to 40 ft. per minute, the maximum being arbitrarily set at 60 ft.

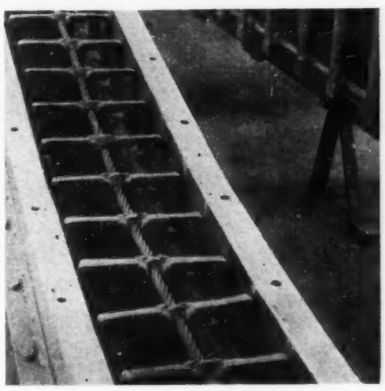

Fig. 2—Conveyor shown in Fig. 1 with cover removed. The return line of flights, inverted, is seen

For conveyors set at angles steeper than the repose angle, including the vertical, the H-shaped flight is used, the rope passing through the center bar.

Power requirements of this system of conveying are relatively small. Only one-third to one-fourth of the power required by a screw conveyor is taken by a Redler unit of the same capacity. The saving as compared with the air transportation system is even greater.

Where there is a horizontal bend in the conveyor line, the rope is attached to the arm of the flight on the inside of the bend.

If many bends are required in a line, a closed tubular trough may be substituted for the rectangular box trough, and circular loops used as flights, the rope being attached to the loop at the bottom. This type of conveyor, however, takes more space, is not as accessible as the rectangular trough with the removable cover, and requires another pipe for the return.

Still another type of conveyor is the "Duo," which is intended for cases where conveying in two directions at one time in one conveyor is desired and the capacity requirements are small. In this, the flight consists of a flat link having no perpendicular arms or members.

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Roan Antelope Adopts Electric Sample Drying

Sample-drying equipment in the Roan Antelope concentrator is incorporated in a special three-door-shelf type Westinghouse electric oven. It handles crushed ore and finely ground mill products. It has been in service one year, without repair or replacement.

The crushed ore is an aggregate sample of ore as delivered from the crushing plant to ball mills for grinding. All but 40 per cent will probably be retained on 0.50 in. It is dried to determine moisture content and resultant dry tons of ore fed to ball mills and the concentrating system. Milling calculations are based on dry tons' input to the concen-

Electric oven used for drying samples at Roan Antelope

trator. Samples containing 3 to 4 per cent moisture are dried in four hours.

The finely ground mill products are samples of pulp after passing through ball mills and range in size from 65 mesh down to colloidal material. They differ in mineral character according to the point of collection in the mill circuit. The samples are collected as relatively dilute pulp of 20 to 25 per cent solids and are first dewatered in vacuum filters which reduce the moisture to 10 to 15 per cent, after which the oven reduces them to practical dryness in about five hours. In this instance, the purpose of drying is to insure dry weight accuracy in subsequent assay and metallurgical calculations.

The oven has an indicator light to insure economy in use of the furnace, the light being lighted only when the oven is heating. Operating temperatures are relatively low, being around 120 deg. C., to avoid burning the sulphur and consequent oxidation of sulphide materials, an important factor in mill accounting and operation. Temperature during this phase of drying is controlled automatically and varies 5 deg. with 120 deg. as the mean. Critical characteristics of the electrical equipment are 550 volts, 3 phase, 62.5 kw., with a maximum temperature of 150 deg. C.

This drying unit has been in operation for one year, and to date no replacements or repairs have been found necessary.

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Mining Industry's Newest Underground Loader Affords Interesting Performance Data

NEWEST among underground loading machines to be introduced to the mining industry is the so-called Eimco-Finlay loader, invented by J. S. Finlay, superintendent of the North Lily Mining Company, near Eureka, Utah, and built by the Eastern Iron & Metal Company, of Salt Lake City. At the time of its introduction it was described in these pages (*E.&M.J.*, April, 1932). Results obtained with it by four of the mining companies who have installed it since then are reported in the following paragraphs. The four companies are: The United States Smelting, Refining & Mining Exploration Company, Utah; the Combined Metals, Inc., Utah; the Alaska Juneau Gold Mining Company, Alaska; and the North Lily Mining Company, Utah.

In the three instances the shovel was used for mucking drifts varying from 5½ to 7 ft. wide and 7 to 8 ft. high, the cars loaded varying from 16 to 27 cu.ft. in capacity and the height of car being 44 to 49 in. above the rail. In two of the installations a decided saving in loading cost is reported, the company in another instance not revealing the information. No special skill was found necessary for operation. The shovel was reported by two of the operators to be flexible in operation in straight drifts, one of the two adding that it worked well on moderate curves and slight grades.

In the United States company's mine, the typical loading place (in which the shovel works) is an untimbered straight drift, 6 ft. wide and 7 ft. high, the bottom being rough and the track of 20-lb. rail, 18-in. gage. The rock loaded is quartzite and limestone, breaking 20 cu.ft. to the ton and to a maximum dimension of 12 to 20 in. Cars, 16 cu.ft. in capacity and 44 in. high above the rail, are supplied nine at a time.

To "load out" a typical heading, requiring the handling of 18 to 21 tons, or 23 to 27 cars, took in the period scrutinized 2 hr. and 30 min. on an average, 40 per cent of which time the shovel was in operation. Tramming, low air pressure, and the occasional lack of air are given as reasons for loss of loading time. Two men compose the heading crew, one handling the shovel and the other the cars. The shovel is moved from face to face by a locomotive, 15 to 30 min. being required for this and 10 to 30 min. for setting up. The shovel has an air engine mounted between the driving axles and geared to them by a roller sprocket chain. Moving the shovel from level to level has likewise proved easy.

Underground loader in action in Golconda Lead workings, Wallace, Idaho

Air pressure is 70 to 75 lb.; and air consumption 60 cu.ft. of free air per minute on the average and 150 cu.ft. maximum. Per ton loaded the consumption is 450 cu.ft. of free air.

During the two months that the shovel has been in service up to the time these facts were determined, maintenance required was limited to renewing the wire rope on the dipper rotator and a few links on the dipper-dumping device. Tonnage loaded in this two months' period was 5,500 to 6,000.

Loading cost for this period was 22.525c. per ton, as compared with 13.475c. for hand shoveling. The 22.525c. per ton charge is distributed thus: Labor, 18.5c.; air, 1.14c.; lubricant, 0.35c.; interest, 0.275c.; and depreciation, 2.26c. Maintenance labor was nominal.

On this basis of cost, the capital expenditure for the shovel should be returned within five months. In calculating the cost, allowance was made for a 1,000-ft. average tram for both machine and hand mucking. Depreciation was figured for a two-year period, which, it is expected, will cover the maintenance cost as well.

In the workings of the Combined Metals Company, the shovel has to handle shale and limestone, running 19 cu.ft. per ton broken, the fragments having a maximum dimension of 19 in. The typical heading is 5½ ft. wide by 7 ft. high. End-dump cars of 1-ton, 19-cu.ft. capacity and 48 in. high are supplied one at a time. The total number (for a given loader) will depend upon tramming conditions. With a 1,000-ft. tram, five cars are used.

Four and a half hours were required to clean out a heading, this yielding 35 1-ton cars of muck. The shovel was in actual operation half of this time. Loss of loading time was charged as follows: 30 per cent, tramming conditions; 20 per cent, cleaning drift from effects of previous blast and extending track and pipe. The management states that when loading conditions are ideal, the cars can be loaded at a rate of 30 tons an hour in the 5½x7-ft. drift. Only one man is needed to handle the shovel, tramming not being considered a part of loading. The company states that an adaptable man can learn to run the shovel in three to five days. The runner is paid miners' wages. Thus far the shovel has been used in only one heading. When it becomes necessary to move it, it will be propelled by its own air engine, air connections to the 2-in. line having been provided at 50- to 100-ft. intervals.

Air pressure is 85 lb. and average air consumption is estimated at 125 cu.ft. of free air per minute.

In the first 60 shifts (30 two-shift days) of operation, to which period these data pertain, the shovel loaded 2,100 tons at a cost of 26c. per ton, compared with an average cost of 34c. for hand loading. A further saving was made by an increase in drifting speed of 2 ft. per shift, won by use of the shovel. The total expense of running the shovel 60 shifts was $550, distributed as follows: Labor, $240; air, $200; lubricants, $5; maintenance, $50, of which $20 was for labor and $30 for supplies; interest, $9; and depreciation (three-year basis), $46. The saving per month is estimated at $400 in overhead, because of the gain of 2 ft. advance per shift and $189 in loading cost, or a total of $589. At an assumed cost of $1,650, the machine will pay for itself in 2.8 months on this basis.

At Alaska Juneau, the typical loading place is a drift 7 ft. wide and 8 ft. high. The material loaded is in general blocky in character and breaks 20 cu.ft. per ton, the fragments frequently being 12 in. in maximum dimension. End-dump roller-bearing 27-cu.ft. cars, 49 in. high above the rail, and running on a 24-in. gage track, are hand-trammed as needed, from a near-by siding. The shovel crew consists of one runner and four trammers. The shovel is moved from face to face by hand, after disconnecting the air power. Air pressure is 85 lb.

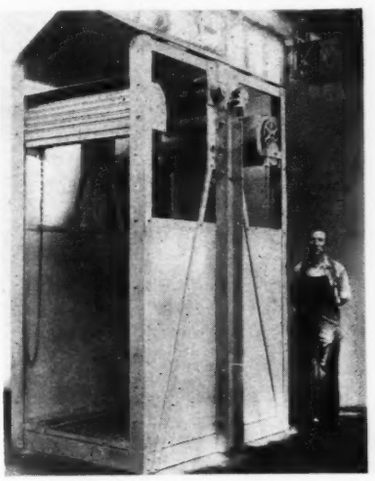
To muck out a heading, yielding 28 cars holding 37 tons, took 2 hr. and 58 min., 68 per cent of which time the shovel was in actual operation. The lost time was caused by various small delays, such as advancing the slide rails, cleaning track, and barring down the face.

During its first two months of service, the shovel loaded 2,600 tons. The only repairs and maintenance required in this time comprised the readjustment of the coil springs, which had become loose, the breaking of the throttle valve stem, and the replacement of the long cam cable. Costs are not available at the moment.

Mr. Finlay, the inventor of the loader, also has supplied data regarding its performance in the North Lily mine. In 15 months' service, one machine has handled 20,000 tons of rock at a total cost of 10 to 12c. per ton, which is approximately 30 per cent of the hand-loading cost. In drifts, where the cost of driving is high because of heat or water, the loader, he says, can be made to pay for itself in two to eight weeks' time. In this mine, the loader is used for driving drifts and crosscuts 4x7 ft. in size and larger. The rock is limestone, shale, and quartzite, running approximately 10 cu.ft. per ton, broken, the fragments of maximum size being about 3x2x1 ft. Cars of 16-cu.ft. capacity, 48 in. high above the rail, are supplied by hand from the motor haulage switch to a portable one-car switch, and thence to loader.

To load the rock in a typical heading takes about 1½ hr. and yields 35 to 45 cars of muck, or 27 to 35 tons. The shovel is in operation about 60 per cent of the total time. The principal cause of loading delay is changing cars. Three men—namely, the runner, a miner, and a trammer—comprise the shovel crew.

Repairs required during the fifteen months were made principally on the engine. By reason of the improvement of the latter, they have since been largely eliminated. To produce the results here set down, according to Mr. Finlay, requires two to four months' practice on the part of the men.



This cage is novel in that it has rolling steel doors

Cage Has Rolling Steel Doors

A type of rolling, steel door that is novel in its application to mine cages is shown in a cut presented herewith. This has been developed by F. W. Collins, of the Joshua Hendy Iron Works, San Francisco. On several occasions in the past, Mr. Collins proposed cage doors of this type to various operators, but not until a condition arose about a year ago, when a certain company found itself in difficulties with the California State Accident Commission over the caging of cars, did he get the opportunity to install them. This company's program called for caging cars about as long as the shaft compartment and for a man riding the cage with the car. The commission insisted that under these conditions the cage must have solid doors and that no swinging door could be employed. When the cage was completed, the doors worked better than Mr. Collins had expected. The doors are very strong, and being spring-balanced, are completely opened or closed by a single pull on the operating chain. The rollers at the top ends are protected by heavy curved plates, so that long timbers or steel can be loaded over them safely. Either door can be quickly removed if necessary for handling very long timbers or rails. The doors were built by Joshua Hendy Iron Works.

INDUSTRIAL NOTES

Birdsboro Steel Foundry & Machine Co., Birdsboro, Pa., has acquired full ownership of C. G. Buchanan Co., 90 West St., New York, designers and builders of crushing and concentrating machinery, and will manage the company directly. In building crushers, the Birdsboro company will adhere to the present design of the Buchanan line, and will handle all designing in its own drafting rooms under the control of a staff of 30 to 35 engineers. Manufacturing will be done at Birdsboro. This acquisition makes the Birdsboro company the only builder of this type of equipment which has its own steel foundry.

Ingersoll-Rand Co., New York, has acquired exclusive rights to market SKF drill steel throughout the world. The steel is sold under the name of IR-SKF.

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BULLETINS

Flotation, Cyanidation. American Cyanamid Co., New York. Technical Paper No. 20, entitled "Recent Developments in Flotation and Cyanidation of Precious Metal Ores." By L. E. Tucker and S. J. Swainson. Pp. 11.

Drying and Grinding. Raymond Bros. Impact Pulverizer Co., Chicago. Bulletin. Raymond kiln mill. Pp. 8.

Industrial Rubber. B. F. Goodrich Rubber Co., Akron, Ohio. Condensed catalog, entitled "Engineering Data, Industrial Rubber Goods."

Steam Plant. Hadfields Ltd., Sheffield, England. Leaflet. Improved nickel-chromium-molybdenum steel for steam-pipe flange bolts.

Plant Ventilation. Coppus Engineering Corp., Worcester, Mass. Bulletin F-320. Coppus Annis dry air filter.

Piping. Taylor Forge & Pipe Works, Chicago. Pamphlet entitled "Taylor Forge," illustrating company products.

Structural Steel. Joseph T. Ryerson & Son, Inc., Chicago. Ryerson Journal and Stock List, Vol. 40, No. 2.

Power Plant. Ready-Power Co., Detroit, Mich. Bulletin 61. Ford-engined ready-power electric plants. Pp. 4.

Blowers. Allis-Chalmers Manufacturing Co., Milwaukee, Wis. Bulletin 1908. Single-stage turbo blowers. Pp. 4.

Boiler Plant. Permutit Co., 440 Fourth Ave., New York. Bulletin on application of zeolite water softeners to treatment of boiler feed water. Pp. 38.

Blowers, Pumps. Roots-Connersville-Wilbraham, Connersville, Ind. Four bulletins: No. 21-B20, No. 21-B21, and No. 21-B23, on applications of Victor-Acme blowers, and No. 21-B22, on an application of R-C-W vacuum pumps. Pp. 2, each.

Screens. Gifford-Wood Co., Hudson, N. Y. Bulletin 123. Mitchell vibrating screens. Pp. 12.

Screens. Newark Wire Cloth Co., Newark, N. J. Catalog No. 32. Wire Cloth Handbook. Pp. 104.

TRENDS AND DEVELOPMENTS IN THE INDUSTRY

Leaders Analyze Metal Situation and Outline Prospects

New York, N. Y. Hope that the speakers would have something encouraging to say regarding the status of copper, lead, silver, and zinc was undoubtedly the reason for the excellent attendance at the regular monthly dinner meeting of the New York Section of the A.I.M.E. on Oct. 19. The speakers who contributed to the symposium on these four metals were Ludwig Vogelstein, vice-president of American Metal, who talked on "Copper"; Clinton H. Crane, president of St. Joseph Lead; C. W. Handy, president of Handy & Harman, whose subject was "Silver"; and R. M. Roosevelt, vice-president of Eagle-Picher Lead, who spoke on "Zinc."

Copper

According to Mr. Vogelstein, the outstanding event of 1932 in copper has been the reimposition of an import duty of 4c. per pound in the United States which became effective June 21. A consequence was that the Empire Conference at Ottawa recommended a duty of 2d. per pound, equivalent at normal exchange to the 4c. duty, on all copper orginating outside the Empire. If and when the latter duty becomes effective, the world's markets will be divided into four units—namely, Russia and Japan; United States; British Empire; and, fourth, the remainder of the world. Aside from Russia and Japan, there has previously been a free international market.

World production of copper is now running at the rate of 930,000 short tons per annum, of which Russia and Japan are producing 124,000 tons, leaving 806,000 tons divided among the other three groups thus: United States custom district, 307,000; British Empire, 212,000; and other countries, 287,000 tons.

At present rate of consumption, the United States will absorb 300,000 tons per annum; the British Empire, 150,000; and all other countries, except Russia and Japan, 380,000; the total being 830,000 short tons. United States consumption is thus seen to be about equal to domestic production (but in this country there is a stock of approximately 650,000 tons of refined copper). British Empire consumption is less than production by 62,000 tons, which must be sold in neutral markets. The rest of the world, except Russia and Japan, is consuming approximately 93,000 tons per year more than it produces. Refined stocks in Europe are estimated at 140,000 tons, and in Great Britain at 23,000 tons.

The world's free markets will hereafter be confined to Germany, France, Belgium, Holland, Austria, and the Succession States and the Scandinavian countries.

This condition will cause a decided change in the marketing of copper, said Mr. Vogelstein. Nobody can say now how it will work. The London standard market may become a paper market. As far as we know, standard copper will be traded as "bonded" copper and will lose contact with the actual market. To make a bad situation worse, Belgian copper entering France is subject to a 2 per cent tax, and copper originating elsewhere is subject to a 4 per cent tax.

The United States, heretofore the leading market, has lost its paramount position. There is no reason why United States producers could not raise the domestic price 1 or 2c. a pound. Their entire present production is needed. The admittedly large stocks of refined copper are well controlled and entirely financed by a group of six or seven corporations which have no intention of dumping their stocks on an unwilling market. The domestic situation is further complicated by the fact that three domestic producers, whose output is about half of domestic production, control fabricating plants representing more than 60 per cent of the domestic consumption, so that the amount of business in the open free market is very small, especially at the present low state of the consuming industry.

The future of the copper market must be viewed in the light of the conclusion that the price movement of metals follows the general business cycle except when the inexorable law of supply and demand is being tampered with. United States' producing capacity is five times as large as its present production; that of Great Britain 2½ times as large as its present production; and that of the world at least three times the present total output. Improvement in business in general and the electrical business in particular, and a sane policy on the part of producers, are needed to restore the world's copper industry to a profitable basis.

In concluding, Mr. Vogelstein emphasized that the fact that consumption in the United States has fallen off much more than in the rest of the world, and that better business is needed in this country. He branded as incorrect the theory that continuation of present low prices of copper will bring about further curtailment. Nobody would dare predict when the present anomalous condition of the copper industry will be solved, he said. Present consumption in the United States is so subnormal that little imagination is required to see that a decided and continued upswing in the use of copper will continue to follow improved conditions in general business. Moreover, economic pressure has resulted in lowering costs, and when the world, especially the United States, emerges from the depression, the copper mines will be in an excellent position to benefit from the economies introduced in the last three years.

Lead

The downward course of conditions in the lead industry, according to Mr. Crane, has brought deliveries of lead for domestic consumption to the low monthly average of 23,000 tons for the first nine months of 1932. Mr. Crane presented figures showing for the period Jan. 1, 1928, to Oct. 1, 1932, average monthly United States deliveries, in total, then subdivided into primary and secondary, as well as figures showing production derived from old battery plates. The large estimate of secondary lead deliveries would be surprising, he said. The increased proportion of total deliveries derived from old battery plates in eight months of 1932 is shown in the table.

Producers have not been able to accommodate themselves with sufficient rapidity to the drastic drop in consumption. Domestic stocks have increased at a rate of about 3,000 tons a month.

Year	Monthly Total (A)	U. S. Lead Deliveries Primary	U. S. Lead Deliveries Secondary	Estimated Monthly Production Derived From Old Battery Plates (B)	Per Cent B of A
1928	79,000	59,000	20,000	11,000	14
1929	82,000	61,000	21,000	11,000	13
1930	65,000	48,000	17,000	9,000	13
1931	48,000	33,000	15,000	9,000	21
1932 (9 mos.)	33,000	22,000	11,000	9,000	28

Mr. Crane explained the relation borne by the custom lead smelters to the market. From the present excess smelting capacity arising from the decrease in mine production has resulted fierce competition between the primary and secondary smelters for the purchase of secondary material. The buyers, although not numerous, have curiously been much less successful in regulating the flow of this lead from the junk man to the smelter than in arranging for an even flow from mine to smelter.

Figures showing the excess of the New York price over the London price

by years since 1923 were presented. The highest differential was 1.8c. a pound, in 1929, and the lowest 1.13c., in April, 1932, showing that the existence of a 2½c. duty has never enabled the United States producer to charge this full differential. At present, the internal competition is keeping the domestic price much below the possible London parity.

A smaller amount of lead is being sold in the United States today than at the beginning of the century, despite the increase in population, said Mr. Crane. He was sure that none present believed the sun was permanently set.

Silver

Like any other commodity, Mr. Handy told the audience, silver is subject to the law of supply and demand, and its price in the long run is determined by the conditions which exist. When the price appears to move contra to basic conditions, as at times, it is due to the market's attempt to anticipate changes in governing conditions. These speculative influences were excluded from further consideration by the speaker in presenting his views as being more or less temporary, and he proceeded to consider the conditions of supply and demand with respect to silver.

Silver supply has two sources: newly mined metal, and bullion resulting from demonetized or debased coinage. Supplies from mine production have been greatly curtailed. The speaker could see no reason for change unless arising from an increased demand for base metals associated with silver. This curtailment is thus a favorable market factor. The situation regarding the second source of supply (melted coinage) is also favorable, Mr. Handy thought. Taken altogther, supplies from this source should not be as great as in the past.

On the demand side, consumption falls into three classes: coinage, the arts and industries, and the Far East. The coinage situation is improving by reason of purchases, or arrangements to purchase, by Germany, Mexico, Cuba, Yugoslavia, Belgium, and Turkey. This revival of coinage from silver is encouraging. Consumption in the arts and industries is showing a decline. This will be small, however, and is a minor factor.

The third class—demand from the Far East—is important. The tremendous shrinkage in demand from India and China is the chief cause of low silver prices, in Mr. Handy's opinion. Instead of importing 75 per cent of the world production of newly mined silver, as ordinarily, these two countries will probably import less than 35 per cent in 1932. Their joint consumption in 1932 Mr. Handy estimated at approximately 50 million ounces, compared with 218½ million in 1929, 217½ in 1930, and 116 million ounces in 1931. This shrinkage is due, broadly speaking, to the severe decline in prices of commodities exported by the two countries, caused by the depression. Because of the decreased ability of the rest of the world to buy the products of India and China, even at low prices (moreover, the two countries are less able to import silver), stocks of silver in the Indian bazaars are relatively large. The statistical position of the market is poor. Likewise in China conditions are said to be "most unkind" to silver. Money instead of flowing through trade channels into the interior, as normally, is accumulating in Shanghai in stocks of already unprecedented size. Importation of sycee from the interior and small coins from Canton is adding to these stocks. Also unfavorable is the Shanghai speculators' oversold gold position, to cover which would require sale of tael exchange with resultant depressing effect on silver prices. A further depressant is the likelihood that Hongkong will not continue to buy silver as cover for its note issue.

Mr. Handy, however, does not expect further collapse in price. The economic vitality of the Chinese and Indian farmers is almost unbelievable. The Far East, moreover, has an affection for silver going back centuries. For the present, he thinks, silver prices will fluctuate within the range of recent months—26c. to 30c.

As to the future, the prospects of silver depend on the economic trend, which Mr. Handy is confident will be upward. He does not expect a return to the days of 1928, nor does he believe that the present stagnation of industry and commerce can continue. Between these two extremes there is room for satisfactory business.

Zinc

Speaking extemporaneously on the zinc situation, and frequently referring to charts, Mr. Roosevelt showed how unfilled orders had risen from 20,000 tons to over 80,000 tons in the spring of 1929, thanks to buying by the steel interests. Stocks had piled up while the industry was, so to speak, getting its breath and becoming aware as to what had happened. They reached their high point about 22 months ago, since when they had actually been reduced somewhat. From that time consumption and production have been closely parallel, the two curves being intertwined "like two snakes," the speaker said.

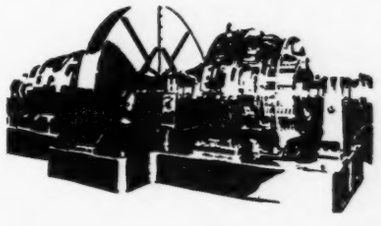
A picture of operations in the zinc industry was given by Mr. Roosevelt. Of the retort plants, nine are running and eleven idle. Of the four electrolytic plants, three are idle and one plant is running. As to this, although the speaker did not mention it, stocks of electrolytic zinc are understood to be very low.

Zinc prices, both in this country and in London, said Mr. Roosevelt, have started to rise. He did not think "all the country could be wrong." The industry in United States is working hard to improve its position. He referred to the movement expressed by the adoption of the "Seal of Quality" for sheets as holding promise for increased consumption.

ARIZONA

United Verde's New Plant. Although production has been curtailed during the past two years at the United Verde mine at Jerome, Ariz., work in the new No. 7 shaft has not been interrupted. This shaft will eventually handle all the hoisting now assigned to the No. 5 and No. 6 shafts. It is now concreted from surface to the 2,700-ft. level, and is being enlarged from the 2,700-ft. level to the 3,000-ft. level, which section will then be concreted. The shaft, 12 ft. 10 in. x 23 ft. 6 in. in the clear, contains two 6 ft. x 6 ft. ore-hoisting compartments, two 5 ft. 6 in. x 12 ft. 10 in. service compartments, and one 4 ft. x 12 ft. 10 in. pipe and manway compartment. The ore-hoisting compartments are inclosed in concrete, but the concrete in the other compartments is in ribs 2 ft. wide, with 4-ft. spaces between. Foundations for the hoist house, hoist, and generator set have been completed and the hoist house is now to be erected.

Electrical equipment was made by Westinghouse. The new hoist was made by Nordberg. Its specifications are as follows: Rope speed 2,500 ft. per minute; 12-ft. diameter drum with a 7-ft. face; load-skip, 14,000 lb.; ore, 15,600 lb.; cable, 23,000 lb.; total load, 52,600 lb.; total weight of hoist, 430,000 lb.; hoisting capacity per day, 4,500 tons; total weight of electrical equipment, 411,500 lb. The hoist will be equipped with a Lilly D controller, automatic accelerator, mechanical retardation, the Ward Leonard control system and an Ilgner fly wheel to stabilize voltage. The hoist will be direct connected to a 2,700-hp., 550-volt, 66-r.p.m., shunt-wound, d.-c. motor. The generator-set flywheel will be 120 in. diameter and will weigh 85,000 lb. Only the ore hoist will be installed at present; that for the service compartments will not be purchased until later. . . . Tombstone Assay Office & Milling has nearly completed erection of a cyanide mill, and the company is beginning to buy ore. V. C. Mellgren is in charge. . . . Bitter Creek Gold Mining has begun operations in the Steeplerock district in New Mexico, 15 miles northeast of Duncan, Ariz. It has purchased the Norman King and Mohawk properties and ten adjoining claims. George H. Currier, of El Paso, is manager, and J. J. Mahon, of Lordsburg, is mine superintendent. . . . At Kirkland, Hassayampa Gold Mining is constructing a transmission line to its property in the Walnut Grove district from the Hammon property. Over this line power will be transmitted for use in a mill the flowsheet for which is being prepared. Other mines in the vicinity are starting operations, including the Ora Bell at Crown King, and the "Thirty-one," Emporia, and Davis-Dunkirk, in the Senator district. . . .



George R. Fansett, of the Arizona Bureau of Mines, and his assistants are conducting a two-day course in placer mining at several towns throughout the state. The course includes instruction in building sluices, panning, the handling of materials, identification of minerals, and methods of testing for minerals, prospectors' diet, clothing, and equipment. The course was first given in Prescott, where more than 200 attended. At Phoenix, where the sessions were also held, more than 250 attended.

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PACIFIC NORTHWEST

Developments at Boise Basin. The new strike of high-grade gold ore at the Gold Hill mine at Quartzburg, in Boise Basin, Idaho, is of importance in demonstrating deep-seated persistence of the vein system. The mine and mill plant of this property was destroyed by forest fire in August, 1931. At that time the shaft had been extended from the 900 to 1,100 level, where a station was being cut at the time the fire occurred. The management has since carried a limited crew on development at the 1,100-ft. horizon and has recently cross-cut a 20-ft. orebody assaying $7.50 and containing 5 ft. on the footwall assaying $16. This is a new oreshoot which apexes close to the 900 level. Orebodies being worked at time of the fire, west of the 1,100 level crosscut, will be developed this winter. The property has a production record of $8,000,000, and development indicates an additional $2,000,000 worth of milling ore above the 1,100-ft. horizon. The gold is as free and amenable to amalgamation at this deep level as it has been in the past. This, the oldest gold mine in Idaho, is owned by Talache Mines; A. H. Burroughs, Jr., is manager. . . . St. Joseph Lead's 200-ton mill at Atlanta, Idaho, has been treating gold ore, said to fluctuate between $12 and $20 per ton, for several months past by amalgamation and flotation. The concentrate, averaging 20 oz. gold per ton, is shipped to Salt Lake City. . . . Golconda, in the Coeur d'Alenes, is developing ore reserves at the rate of three tons for each ton mined in the course of development work. The Mayflower vein, cut on the 1,000 level, shows a width estimated at 8 ft., with content of 14 per cent lead, 10 oz. silver, and 5 per cent zinc. . . . Gold Producers Association, operating 16 miles from Warren, in the Big Creek Boise basin, recently started its 125-ton mill. Gold is being recovered by amalgamation, and a concentrate is produced with an assay value of about $500 per ton. . . . Several hundred miners have returned to work at the Morning and Page mines, in the Coeur d'Alenes, the men having voted to accept an employment proposition offered by the owners. Nearly 2,000 persons will benefit, as most of the men have families. . . . Pend Oreille Mines & Metals is expected to resume production at its Metaline properties at an early date. The balance sheet of the company shows current assets of $180,418; current liabilities, $23,905. Between Jan. 1, 1931, and April 30, 1932, 112,936 tons of ore was handled with a development and mining cost of $1.01 per ton. Milling costs were given at 57c. per ton, and all other operating costs at 0.351c. per ton; total, $1.931 per ton. General expense, legal costs, transfer agency, fees, insurance, taxes, and incidental charges added 0.366c. to the foregoing costs, and brought the all-cost per ton to $2.297. . . . A mining convention will be held in Spokane in December, according to plans made at a meeting recently of the Northwest Mining Association and the Washington Natural Re-Resources Association. . . . The Lorentsen centrifugal amalgamation process is to be used by the Chirikof Gold-Platinum company for the recovery of precious metal in the beach sands of Chirikof, Kodiak, and Trinity islands, off the coast of Alaska. Equipment is being supplied by Gold Savers, Ltd., Vancouver, B. C. . . . The Alaska branch of the U. S. Geological Survey is continuing the work with the Alaskan Railroad, to develop the mineral resources of the railroad belt. Ralph Tuck, of the U. S. Geological Survey, stationed at Anchorage, Alaska, is acting as mineral adviser to the manager of the railroad.

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COPPER COUNTRY

Passing of a Consistent Producer. Mohawk Mining, after many years of successful operation, is in process of liquidation. This action was hastened by the caving of No. 6 shaft, the only productive unit, in the bottom levels. This shaft was assumed to have several months more of life, but physical conditions made continuance impossible. It was rich in ore; copper was being produced for less than 5c. per pound when the shaft closed. Machinery at the mill is being dismantled. Although large sales of copper have been made during the last few months, a considerable stock of mineral remains in storage at Gay, equivalent to 25,000,000 lb. of refined copper. This will be treated at the Michigan smelter at Houghton as a market is found for the metal . . . What disposition will be made of the Michigan mine property in Ontonagon County, owned by Mohawk, remains to be learned. A good showing of copper was in evidence when operations ceased, but the Mohawk company is not expected to continue development. The property eventually may be taken over by one of the other companies in the district. Michigan has an extensive territory and is believed to have splendid possibilities . . . Mohawk, one of the best producers in the Michigan district, was well managed and costs were low. Naturally, its closing is a source of much regret to the people of the Copper Country. It has disbursed to shareholders about $115,000,000, paying its first dividend in January, 1906, and missing only in two years since then, in 1921 and 1924. The mining companies in Houghton county, Michigan copper district, pay the bulk of the taxes. A reduction of $458,417.87, or 27 per cent, has been made in 1933 taxes as compared with the tax levy of 1932. This will effect a considerable saving for the companies. The total levies in Houghton County for 1933 are $1,238,214.

Copper Company Invests in Canada. Arcadian Consolidated Mining has acquired claims in the Kirkland Lake gold district of Ontario, and development work has already started. President Robert H. Shields, of Houghton, Mich., says: "Last August I visited this district, accompanied by our mining engineer, Herman W. Fesing, and we both became favorably impressed. We acquired certain claims, which were selected and approved by a Canadian mining engineer who has had years of experience along that line and in that district; in fact, no less than three paying gold mines are now operating on claims which he recommended, and this same engineer will have charge of the development work for the Arcadian. On account of the depressed conditions in copper, development work on the Arcadian property in Michigan was suspended two years ago, and all efforts will now be concentrated in its holdings in the Kirkland Lake gold district. . . . A change in crushing practice will be tried out in the Michigan copper district. Experimentation has been done on amygdaloid copper rock with cone crushers, and the results are said to be promising. The principal difficulty has been in the malleability of the copper, the large and small masses merely changing in form while the rock is crushed. By sorting out these pieces beforehand, however, and using a series of cones in closed circuit, a method may be adopted that is more efficient and economical than the present steam stamps.

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COLORADO

Copper Company Turns to Gold. Following an examination, North Butte Mining has secured an optional interest in Unity Gold Corporation, organized to acquire and reopen the Katinka group of claims between and adjoining the Mary McKinney and El Paso mines, in the Cripple Creek district of Colorado, under a favorable lease and option. The workings of both the Mary McKinney and El Paso mines extend to the end-line boundaries of the Katinka group. The property is on the southwesterly contact between the Cripple Creek volcanic breccia and the granite surrounding the volcanic plug, with at least one strong phonolite dike traversing the property in a northerly and southerly direction. The gold ores of Cripple Creek are usually closely associated with phonolite or basalt dikes, and occur in the southerly portion of the district in veins which have a general north-and-south strike. The southerly portion of the district has been most productive. Reopening of the mine will entail but moderate expense, as there is a well-timbered vertical shaft 1,200 ft. deep on the property, an electric power line close by, and railroad tracks conveniently located below the collar of the shaft at proper elevation for the economical loading of ore and transportation to the Golden Cycle mill, at Colorado City, where custom ores from the Cripple Creek district are treated. The operations of Unity Gold Corporation are being carried on under the supervision of Evan J. Williams, who has had many years' local experience.

IRON COUNTRY

Mines Provide Employment. The ranges are in a more optimistic frame of mind this winter, as the operating companies have announced the opening of most of the underground mines on a two- and four-day-a-week basis, allowing all the old men to put in at least eight days a month, which will keep the average family from going hungry without aid from charity. Recently, the companies unanimously demanded a 20 per cent cut in ore valuations. This matter is being fought out before the tax commission. With all other types of property suffering at least a 50 per cent reduction, the request for a 20 per cent cut is a modest one.

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CALIFORNIA

Widespread Activity in Gold Mining. Valley View Rand Mining, operating a property near the Yellow Aster, at Randsburg, has developed sufficient reserves of ore to justify erection of a 100-ton mill, for the erection of which arrangements are being made. . . . Production operations are expected to commence in December at the King Solomon group of mines, near Yreka, on the Salmon River. An amalgamation mill is nearing completion. . . . Paymaster Mining, formerly Broken Hills Silver Corporation, operating properties that include the Paymaster mine, near Nevada City, now has a mill and flotation plant in operation, of a capacity of 50 tons daily. . . . The Chollar, Savage and Gold & Curry interests, engaged in the development of a section of the Comstock Lode at American Flat, recently added ten placer and four mining lode claims to their holdings. . . . Idaho-Maryland, at Grass Valley, is rehabilitating its surface plant and retimbering part of its main shaft, with plans under way for an increase in ore-hoisting facilities. Conditions in the mine are said to justify consideration of a further investment in ore-reduction and metallurgical plant. . . . Central Eureka, in Amador County, reports extension of the orebody to the 2,200 level, indicating a persistence that materially adds to reserves. . . . Aladdin Mining, operating the Bright Star claims, near Mokelumne Hill, announces that milling will be commenced within the next few weeks. . . . Western Precipitation, of Los Angeles, is sponsoring the development of what is expected to prove one of the largest hydraulic mining operations in California—at Howland Flat, Sierra County. Construction is proceeding according to schedule, with plans laid for the storage of débris against the Bullard's Bar Dam. G. M. Colvocoresses is consulting engineer. . . . The Southern Industrial and Mineral Committees of the California State Chamber of Commerce, having been advised by officials of the State Industrial Accident Commission that the general application of safety practices may result in a reduction of 25 per cent of the cost of compensation insurance, is taking steps to insist that operators in San Diego and Imperial counties adopt and use the code of mine safety practices recommended by the Chamber. . . . Resumption of operations at the Plumas Eureka placer mine in Plumas County is scheduled for the near future, as soon as construction work now under way is completed. . . . Deep-water divers are aiding in the recovery of gold from the Tuolumne River near Jacksonville. Drag-line excavators and gravel pumps are being used.

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UTAH

Aiding Mining Development. The geology of the Stockton and Fairfield quadrangles, in the southern part of the Oquirrh Range, Utah, has been studied by the U. S. Geological Survey, the results appearing as Professional Paper No. 173. The chief orebodies are bedded replacement deposits in limestone and have been obviously localized by the intersections of fissures and small faults with certain favorable limestones. The chemical composition of these limestones appears not to have been a factor in controlling ore deposition, and the causes of the selection of certain beds rather than other apparently similar beds remains obscure. However, the systemmatic study of the geology has permitted the stratigraphic relations of the several productive limestones to be determined and thus serve as a guide to further exploration for ore. The orebodies of Stockton do not occur in the same limestone beds as the great replacement bodies of the Bingham district, as had been commonly believed, but are several thousand feet lower, stratigraphically. Consideration of the stratigraphy and structure of the range as shown in maps should assist the mining developments in the region, as they facilitate the recognition of the strata that have proved productive. Alteration of the rocks in the neighborhood of the orebodies has been studied in detail, and the features thus established as characteristic of the metalliferous deposits have been pointed out.

Mining Developments. Sinking of a winze below the 1,350 level of Eureka Standard of Tintic Standard has been started, in accordance with the policy of James W. Wade, general manager, to do development work mainly in the winter, when the mine is cooler, and confine summer activity to shipping. During the winter, however, shipping will be continued at the normal rate. Pumping equipment, handling hot water struck at depth, is operating efficiently. . . . Tintic Standard Mining is making a geological survey of claims near the Rio Tinto property at Mountain City, Nev., recently acquired by International Smelting. Additional property, the Hopewell group, has been acquired on the west fork of California Creek, 10 miles northeast of the Elko County camp. . . . Pumping equipment has been installed in the Rio Tinto property of International Smelting at Mountain City, to continue sinking of the shaft below the 257 level. The ore, from 30 to 50 per cent copper, was first struck at the 227 level, and has been continuous from this depth to the lowest point reached in the mine, now the 275 level in the shaft. A drift on the 250 level followed ore for 125 ft.

CANADA

Tyrrell Property Is Promising. The first underground work in the Swayze gold area, 125 miles nothwest of Sudbury, has been commenced. An oil-driven mining plant has been shipped to the Kenty property, with which to sink to 500 ft. and explore the vein system down to this level. Surface showings exposed during the past year under a heavy overburden indicate a complicated system throughout a length of 5,000 ft. and a width of several hundred feet. Abundance of gold is visible in places, and assays are so erratic as to make channel sampling difficult. The property is owned by J. B. Tyrrell and associates, who have provided the funds for the present underground program. E. T. Corkill is in charge of operations. A good deal of surface prospecting has been done in this area during recent months, indicating a large gold-bearing area. . . . Granada Mines, in the Rouyn district, is making a small profit from $10 ore, and searching for larger deposits with more consistent values at the 1,075-ft. level. Electric power, provided last January, permitted tonnage to be raised from 75 tons to 100 tons daily. Originally the mill made a blanket concentrate, which was sold to the Noranda smelter. Provision has now been made for cyaniding this concentrate, which gives a 98 per cent recovery at one-fifth of the former smelter charge. The operating profit for the last fiscal year was $65,000. . . . The end of a prominent silver producer is signalized in the report of Keeley Silver Mines, whose output from 1913 to October, 1931, when it was closed, was 12,182,952 oz. silver and 1,562,381 lb. cobalt. A considerable part of the profits was used to search for a new mine, without success, and much of the income from the present surplus of $900,000 is being used for the same purpose.

Porcupine Mine Progresses. Wright-Hargreaves has ordered, from an English firm, a new hoist for its central shaft, to facilitate development between the 3,000- and 4,000-ft. levels, which now has been commenced. Diamond drilling has indicated good values and volume of ore down to 3,800 ft. Lateral development down to 3,000 ft. during recent months has increased the ore reserves materially, and has raised the average grade above the $13 that represents ore now going to the mill. Although this mine can hardly hope to rival its neighbor on the west, Lake Shore, it has become one of the leading gold mines of Ontario, its September output ranking fifth in the province, after Lake Shore, Hollinger, Tech-Hughes, and McIntyre. . . . Mine and mill equipment for the Island Lake gold property of Ventures, Ltd., has been assembled at the north end of Lake Winnipeg, ready for overland shipment by winter road to this isolated locality, reached only by airplane during the summer. The 50-ton mill, expected to be in operation in February or March, will provide a modest income in addition to determining definitely the value of the deposits, most of which lie beneath the lake. At God's Lake, near by, the prospect held jointly by Robert Jowsey and Coniagas Mines is responding well to surface development.

Siscoe to Adopt Flotation. J. M. Forbes, general manager of Siscoe Gold, reports that the main shaft has been completed to 1,000 ft., and levels are being developed at 725 and 850 ft., on the former of which ore has now been encountered. The development of lower levels will be guided by results on these two. Development on the upper levels has been so satisfactory that no difficulty will be experienced in providing the 225 tons a day required for the enlarged mill next February. Tests indicate that the present 20-cent tailing may be reduced to 10c. or less, even from ore running $20 or better. The mill at present obtains its gold from a trap, corduroy blankets, and small cyanide plant. A flotation unit is being added after the blankets, which will give a tailing with negligible gold. The present cyanide plant will treat flotation concentrate from an estimated mine tonnage of 300 tons. Additional power is being provided from a 675-hp. diesel-electric unit.

BRITISH COLUMBIA

Cyanide-Flotation Activity. Production of the Consolidated company during the third quarter of 1932 shows that gold output increased from 6,065 oz. to 8,837 oz. Output for the first nine months of this year indicates a sharp increase in gold of 5,918 oz. . . . Meridian Mining has been organized to reopen and operate as a unit the Oyster-Criterion, Eva, and other formerly productive gold properties near Camborne, in the Lardeau district. The consulting engineer is A. G. Langley, of Vancouver, who reported favorably on the possibilities of the development of a considerable tonnage of low-grade ore, which could be mined and milled profitably at a cost of about $3 a ton. A development program has therefore been planned in the expectation of proving sufficient ore to justify the modernization of the existing Oyster-Criterion mill to employ cyanidation and flotation. On the Eva are two quartz veins carrying siderite and sulphides, together with free gold. These veins, varying from a few inches to many feet wide, lie in and along two fault planes connected by numerous veins and stringers. From this property some years ago 30,595 tons was mined and milled, yielding an average of $4.86 a ton. . . . The new mill of Cariboo Gold Quartz, near Barkerville, is expected to be in operation in November. The metallurgical problem is a simple one and calls merely for all cyanidation. The operation will also be economical. To secure maximum recovery it will not be necessary to grind finer than eighty mesh. . . . The new counter-current decantation unit of the Pioneer mill is now in operation, giving a present capacity of about 250 tons a day, which will be increased to over 300 tons. The mill is now being run on ore from development and on the excess from shrinkage-stoping operations between the seventh and eighth levels. . . . Erection of a mill on the property of the Big Slide company is being considered. This was one of the earliest gold mines in the Province to be worked. After being abandoned for many years it is being re-explored, with encouraging results.

AUSTRALASIA

Victoria Gold Output Up. Bendigo continues to be the scene of great mining activity, the output for a recent month being 2,231 oz., compared with 1,290 oz. for the same period in 1931. Prospectors at work under government assistance and guidance have recovered 5,062 oz. during this year. The new saddle reef below the 1,620 ft. level of the Hercules mine is showing high-grade ore, 2,028 oz. being obtained so far from 1,153 tons. Many mines are being prepared for production, and output from the Bendigo field should still further increase. . . . Recent gold-mining companies incorporated in Victoria include A1 Consolidated, to acquire and develop mines at Woods Point; Mount Delegate Gold, to open up a property at Bendoc; Alison Mining to work auriferous antimony lodes at South Costerfield; Neangar Gold Mining, to unwater and further develop a mine at Eaglehawk, near Bendigo; Corindhap Hydraulic Sluicing, to acquire 146 acres of wash at Rokewood; New Glenfine Gold, to acquire alluvial leases near Ballarat. . . . Numerous large nuggets have been found on Victorian gold fields recently. At Beaufort, a nugget weighing 45 oz. was found in alluvial wash at a depth of 20 ft.; near Castlemaine, a slug weighing 21 oz. was located just under the surface. At Dunolly, a specimen valued at £180 and weighing 26 oz. was found 6 in. below the ground. These finds have greatly stimulated prospecting.

Technical Progress in South Australia. During the last decade, Broken Hill Associated Smelters, which treats at Port Pirie, South Australia, the output of lead concentrate from Broken Hill, recovered an average of 146,000 tons of lead, 8 million oz. silver, and 4,340 oz. gold annually. Capital expenditure for the period totaled £1,500,000, mostly spent in installing a telpher handling system, in improving working conditions, and in plant research. This research has resulted in perfecting a method of continuous desilverizing and degolding of lead bullion, in place of the intermittent Parkes system. The slag dump at the Smelters contains nearly 600,000 tons of zinc. When the price of zinc rises sufficiently, a plant to treat the accumulated and current slag by fuming will be installed. . . . Australian Radium Corporation, which owns the radium leases at Mount Painter, South Australia, and a treatment plant near Adelaide, has formed Australian Radium Products, with a capital of £300,000. Recent development work on the surface exposures of the torbernite-autunite lodes at Mount Painter have been promising, and seven formations have been tested on fourteen leases. The ore is to be concentrated at the mine and the concentrates dispatched to Melbourne for treatment in a new plant, with an output of 2 grams radium monthly. . . . Much interest was aroused as the result of a discussion at the recent Broken Hill meeting of the Australasian Institute of Mining & Metallurgy upon the future of the Broken Hill field. Andrew Fairweather, manager of Broken Hill South, estimated a life of 20 years on the basis of 20 million tons of ore to be mined. From a saddle-shaped lode three miles long and 500 ft. wide, 4 million tons, valued at £157 million, has been mined since 1883. The South Mine is expected to be the first of the three major mines to close down, as the ore is tending to pitch out of the lease, in the north; and Zinc Corporation exploratory work may, as in the past, prove further orebodies. In 1913, the mines employed 10,000 men. Today the figure is 3,200.

Re-treatment Plant Successful. Golden Horseshoe New, at Kalgoorlie, W. A., has re-treated about a million tons, or one-third, of the dump of tailing of the old Golden Horseshoe mine. The cyanide plant is highly efficient, and costs are less than 2/- per ton. With a view to increasing the life of the company, effort is being made to acquire additional tailing for re-treatment. . . . Wiluna treated 28,437 tons during August for a yield valued at £60,737 without the English premium. Working costs at £30,294 gave a surplus of over £30,000. During the last few months the company has been experimenting with new flotation reagents, and an increase in recovery is reported. Building extensions for two additional diesel engines now on order are in hand. . . . The 250-ton mill at Mount Coolon, Queensland, will be in operation during November. Since the prospectus of the company was issued in October, 1931, further work has proved additional ore of an average value of 90/- per ton above the 300-ft. level, and 55/- per ton below it. At the commencement of operations the ore milled will assay 69/- per ton. Recovery is expected to be 63/- per ton. . . . The Federal Government suspended on Sept. 30 the bounty of 10/- per oz. on new gold. The price of fine gold in Australia is now £7/6/- per ounce. When it falls to £5/10/- per ounce, the bounty will again be paid. . . . Bululo Gold Dredging, operating in New Guinea, recovered 13,466 oz. of gold from 612,000 cu.yd. for the twelve weeks ended Sept. 9. The dredging is now being confined to the eastern side of the property, outside the limits of the original area. Because of low dredging cost, this low-grade wash can be profitably treated. The second dredge, with a capacity of 200,000 cu.yd. monthly, is now on operation. The third dredge, which will work on the Bulowat area, is in course of erection; order has been placed in Sydney for a fourth. These dredges will give a turnover of 800,000 cu.yd. monthly.

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NEW ZEALAND

Output and Prospects Favorable. Gold mining is active in New Zealand. Wetherstones Gold Mining has been formed to develop an alluvial property in the Tuapeka district. A total of thirteen new gold-mining companies has been formed in New Zealand this year, and the output from the mines in operation has been substantially increased. Gold exported from New Zealand for the seven months ended July was valued at £555,379, compared with £324,161 for the similar period last year.

AFRICA

Rhodesian Copper Cost. Rhokana Corporation, and Roan Antelope Copper are delivering blister copper, 99.60 per cent fine, at European ports at a cost of 3½c. per pound, including administration and all other expenses except interest and depreciation. When the work of equipping the mines began, the estimates of production costs ranged from 7½c. to 8½c. Private estimates affirmed that costs as low as 6c. might be achieved. These estimates were based on the known cost of steel, powder, coal, timber, labor, and the consumption of these per ton of ore handled by other companies everywhere. The land and water freights were known also, as were port charges, insurance, and every other conceivable factor. Since operations began, the land and water freight rates have been reduced, though not greatly, and the per ton price of coal also; but these items account for only a small part of the saving in costs. The way the ore drills, breaks, and crushes has made consumption of steel, powder, and power much smaller than anyone anticipated. From the original estimates a saving in mining alone equivalent to about 2c. per pound of copper has been effected. Coarse- and fine-crushing equipment has a far greater efficiency than expected, and consumes much less power. There is also a considerable saving from expected smelting costs. . . . Except in operations where byproduct metals recovered were figured as an offset against expenses, copper probably never at any time previously has been produced and delivered to consumers for as low a cost as that of the Rhodesian companies at present. The costs are now near what may be expected as a minimum, though some items may still be reduced slightly; and operations at capacity will spread overhead expenses more thinly over the cost per pound of copper. . . . Sir Auckland Geddes stated at the recent meeting of Rhokana Corporation stockholders that the company's cost of delivering blister copper in London or European ports, including every expense except debenture interest and depreciation, was £23.17 per long ton in July and £22.52 in August. Taking exchange at $3.48, these costs were the equivalent of 3.59c. and 3.50c. per pound, respectively. He said a further reduction of costs was confidently expected by the company's technical staff. . . . Recent tests have proved that the small amount of cobalt in N'Kana ore can be separated and recovered in the form of a high-grade iron-cobalt alloy, which should find a ready and profitable market.

Dust Abatement Problems. The annual report of the government mining engineer makes mention that the control over machine rock drills on the miners' phthisis schedule continues, but that experience strengthens the opinion held that, despite care in design, the rock drill fitted with an internal water tube requires an impracticable amount of care in maintenance and use to secure under working conditions the low dust release shown in special tests.

Tin Production Resumed. As a result of the rise in the price of tin, Geomines has resumed production of cassiterite in Katanga. In September it produced 65 tons of 70 per cent concentrate. Symetain has started working its alluvial tin fields in Maniema, Eastern Belgian Congo.

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EUROPE

Copper Deposit Discovered. The Royal Geological Survey reports receipt of information to the effect that an important find of copper ore, similar in character to the ore in the Boliden field, has been made in the Skellefte district of northern Sweden. The Survey is making a detailed study of the mineral resources of the country, but development work on new deposits will not be undertaken until market conditions improve.

Diamond Production Control. For several weeks the diamond market has been increasing in strength at Antwerp and Amsterdam. World diamond producers met at Brussels recently with representatives of the Diamond Corporation, of London. It was agreed the Diamond Corporation would continue to be the sole sales agent; but the Antwerp diamond cutters, desirous that Forminière and its subsidiaries sell to them direct the diamonds produced in the Belgian Congo and in Portuguese West Africa, have been assured that their needs would be regularly supplied.

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MEXICO

New Gold Plants Planned. Development work at La Escondida mines, near Quiriego, Sonora, operated by A. J. Wallmark and associates, has revealed several new orebodies, from which eleven carloads of ore has been shipped to El Paso, Douglas, and Hayden. The company intends to erect a treatment plant this year. . . . Low market prices for basic metals precludes immediate prospect for a betterment of conditions in Mexican mining, in the opinion of the mines department of the Ministry of Industry, Commerce and Labor. The department sees no benefit to copper producers from increased prices in American markets, as high import levies almost exclude Mexican metal from the United States, and scant demand exists in Europe. The drop in lead prices and the failure of silver quotations to improve do not augur favorably for an immediate improvement in Mexican mining conditions. . . . Machinery has been installed and engineers have been assigned to determine the extent of gold placer deposits at La Maria, Santo Domingo, and Guadalupe, Chihuahua, by Santo Domingo Gold Mining, an American enterprise registered in Mexico, announces the Ministry of Industry, Commerce and Labor. Between Aug. 1, 1931, and June 15 last, 3,921 concessions for works on mining lands were granted companies and individuals in Mexico. This was an increase of 1,353 concessions over the corresponding 1930-31 period. Of the concessions granted during the 1931-32 period, 2,180 were for exploitation and 1,742 for prospecting. In regard to states, most concessions were granted as follows during the period: Chihuahua, 681; Sonora, 596; Durango, 439; Guerrero, 349, and Sinaloa, 277. The Ministry's statement shows that eight mining companies suspended work on their properties and that thirteen others reduced working staffs and salaries. . . . To provide facilities for obtaining gold sufficient for minting purposes, and to afford further reserves for bank notes, in Mexico's first steps toward re-establishing the gold standard abandoned under a low of July 25, 1931, the government has declared as part of the national mineral reserves two gold-silver fields known as San Miguel Peras and Cerro Colorado, in Oaxaca, and appointed a committee of engineers to ascertain the gold-bearing possibilities of fields that have been abandoned or are inactive throughout Mexico. Mining rights held by individuals or companies to such tracts will be respected, the government announces, and all possible steps will be taken to prevent irregularities in the exploitations of such fields. The Ministry of Finance announces that it bought the total national gold production during September and October, amounting to 2,200 kilos. . . . Under arrangements made between it and its employees, Peñoles Mining has temporarily suspended operations at its smelting plant in Torreon, Coahuila, because of unprofitable operations. During the shutdown, ores are being handled by the company's smelter at Monterrey, Nuevo Leon. Workers of the Torreon plant were indemnified with 30 days' pay each. . . . The mill of Namaquipa Mining, about 50 miles from Tecolote, Chihuahua, is being reconstructed, and the enterprise is shipping about 600 tons ore monthly to Chihuahua City. . . . A mill is being built on properties owned by Monte Verde Mining at La Mina San Joe, Chihuahua state. . . . The Churunibabi Mines at Nacozari, Sonora, operated by Moctezuma Copper until silver prices reached the 36c. level, and then leased to Samuel Clinch and associates, is shipping 900 metric tons of ore a month to the United States.

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CENTRAL AMERICA

Mocupia Company Reduces Costs. Cie. Française de la Mocupia, which owns 20,000 shares in the New Goldfields of Venezuela, reports that a second Dorr thickener, a second Oliver filter, portable Atlas-Dorman air compressors, and rock drills have been sent to the mine. A second 80-hp. Winterthur motor, with wood-gas producer plant, is in course of delivery. Tonnage treated in July totaled 1,610; in August, 1,952; and in September, 1,970 tons. Yield in the third quarter of 1932 totaled 60 kilos of gold, valued at slightly over one million francs. Net costs per ton, which were 10.7 grams for the first six months of 1932 (leaving 400,000 francs profit of exploitation), have been reduced, by increasing the tonnage treated, to 10 grams gold per ton for July, and 9 grams for August.

Daily and Average Monthly Prices of Metals

October, 1932

United States Market

Oct.	Electrolytic Copper: Domestic Refinery	Electrolytic Copper: Export Refinery	Straits Tin: New York	Lead: New York	Lead: St. Louis	Zinc: St. Louis
1	6.025	5.625	24.500	3.000@3.100	2.950@3.000	3.100
3	6.025	5.550	24.200	3.100@3.150	3.000	3.050@3.100
4	6.025	5.450@5.500	24.500	3.250	3.100@3.150	3.050@3.100
5	6.025	5.500	24.350	3.250	3.100	3.050@3.100
6	6.025	5.425	24.100	3.250	3.100	3.050@3.100
7	6.025	5.400	24.000	3.150	3.000	3.000@3.100
8	6.025	5.425	24.000	3.150	3.000	3.000@3.100
10	6.025	5.350	23.900	3.000	2.900	3.000@3.100
11	6.025	5.275	24.100	3.025	2.900	3.000@3.050
12	Holiday	5.325		Holiday		
13	6.025	5.350	24.250	3.000	2.900	3.000@3.050
14	6.025	5.275	24.250	3.000	2.900@2.950	3.000
15	6.025	5.275	24.300	3.000	2.900@2.925	3.000
17	6.025	5.325	24.350	3.000	2.900@2.925	3.000@3.050
18	6.025	5.325	24.250	3.000@3.050	2.900	3.000@3.050
19	6.025	5.325	24.050	3.000@3.050	2.900	3.000@3.050
20	5.775	5.225	24.000	3.000	2.900	3.000
21	5.525	5.175	23.900	3.000	2.875@2.900	3.000
22	5.525	5.175	23.850	3.000	2.875@2.900	3.000
24	5.400	5.150	23.450	3.000	2.875@2.900	3.000
25	5.275	5.075	23.450	3.000	2.900	3.000
26	5.150	5.025	23.250	3.000	2.875@2.900	3.000
27	5.025@5.150	4.900	23.250	3.000	2.900	3.000
28	5.025	4.875	23.300	3.000	2.875@2.900	3.000
29	5.025@5.150	4.900	23.500	3.000	2.900	3.000
31	5.025@5.150	4.925	23.350	3.000	2.875@2.900	3.000
Av for Month	5.733	5.256	23.936	3.052	2.939	3.027
Oct.			**Averages for Week**			
5	6.025	5.579	24.442	3.125	2.996	3.113
12	6.025	5.367	24.020	3.115	2.980	3.050
19	6.025	5.313	24.242	3.008	2.908	3.017
26	5.442	5.138	23.650	3.000	2.892	3.000
Oct.			**Calendar Week Averages**			
1	6.025	5.675	24.588	3.096	2.958	3.198
8	6.025	5.463	24.192	3.196	3.054	3.067
15	6.025	5.308	24.160	3.005	2.908	3.020
22	5.817	5.258	24.067	3.008	2.898	3.013
29	5.171	4.988	23.367	3.000	2.894	3.000

Silver, Gold, and Sterling Exchange New York and London

Oct.	Sterling Exchange: "Checks"	Sterling Exchange: "90-Day Demand"	Silver: New York	Silver: London	Gold: London
1	3.455000	3.457500	27.250	17.6875	.....
3	3.453750	3.456250	27.375	17.6875	119s 3d
4	3.450000	3.453750	27.375	17.6875	119s 4d
5	3.452500	3.455000	27.375	17.8750	119s 4d
6	3.451250	3.453750	27.250	17.6875	119s 5d
7	3.452500	3.455000	27.375	17.6875	119s 6d
8	3.457500	3.461250	27.375	17.7500	.....
10	3.453750	3.457500	27.500	17.7500	119s 7d
11	3.440000	3.443750	27.375	17.6875	119s 6d
12	Holiday			17.7500	119s 9d
13	3.441250	3.443750	27.500	17.8125	119s 9d
14	3.441250	3.442500	27.500	17.6875	119s 9d
15	3.443750	3.445000	27.375	17.6875	.....
17	3.445000	3.446250	27.375	17.6875	119s10d
18	3.407500	3.408750	27.375	17.7500	120s
19	3.390000	3.390000	27.375	17.8750	122s 2d
20	3.396250	3.395000	27.375	17.8750	121s10½d
21	3.395000	3.391250	27.000	17.7500	121s 7d
22	3.392500	3.387500	27.000	17.6875	.....
24	3.326250	3.316250	27.125	18.0000	122s11½d
25	3.300000	3.297500	26.750	18.0000	125s 4½d
26	3.302500	3.302500	26.625	17.7500	124s 3d
27	3.282500	3.281250	26.625	18.1250	125s 8d
28	3.281250	3.281250	26.750	17.9375	125s 8d
29	3.281250	3.281250	27.000	18.1250	.....
31	3.282500	3.285000	26.875	18.1250	125s 5d
Av. for Month	339.500	.....	27.195	17.813	.....
Oct.		**Averages for Week**			
5	3.45229	.....	27.375	.....	.....
12	3.45100	.....	27.375	.....	.....
19	3.42813	.....	27.417	.....	.....
26	3.35208	.....	26.979	.....	.....

Calendar week averages: New York Silver, Oct., 1st, 27.417; 8th, 27.354; 15th, 27.450; 22nd, 27.250; 29th, 26.813.

London Market

October	Copper: Standard Spot	Copper: Standard 3 Mo.	Copper: Electrolytic (Bid)	Tin: Spot	Tin: 3 Mo.	Lead: Spot	Lead: 3 Mo.	Zinc: Spot	Zinc: 3 Mo.
3	33.6875	33.8125	38.000	150.000	151.625	12.5625	12.8125	15.1250	15.2500
4	33.2500	33.3125	37.500	151.250	152.500	12.4375	12.6875	15.0625	15.2500
5	33.6250	33.6875	37.750	151.250	152.625	12.6875	12.8750	15.2500	15.4375
6	32.6875	32.8125	37.250	149.125	150.500	12.4375	12.6875	14.8750	15.1250
7	32.5000	32.6250	36.750	148.875	150.375	12.2500	12.4375	14.6875	15.0000
10	31.7500	31.9375	36.000	148.000	149.375	11.8750	12.1250	14.5000	14.8125
11	31.7500	31.8750	36.000	149.375	150.500	11.8125	12.1250	14.6250	15.0000
12	32.4375	32.6250	36.500	151.500	152.625	12.1250	12.3125	14.8750	15.2500
13	32.1250	32.3125	36.250	151.375	152.625	11.9375	12.1250	14.7500	15.0000
14	31.5000	31.6250	36.000	149.750	150.875	11.6875	11.8125	14.3750	14.6875
17	31.8125	32.0000	36.250	151.500	152.125	11.7500	11.8750	14.5000	14.8125
18	31.6250	31.8125	36.250	153.000	153.375	11.5625	11.7500	14.5000	14.7500
19	32.3125	32.5000	36.500	153.125	153.375	11.9375	12.1250	14.9375	15.3750
20	32.0000	32.2500	36.000	151.875	152.375	11.7500	11.9375	14.8750	15.1875
21	31.7500	32.0000	35.750	151.750	152.000	11.8125	12.0000	14.8750	15.1875
24	31.3750	31.5000	35.500	151.250	151.625	11.7500	11.9375	14.8750	15.1250
25	31.5625	31.6875	35.500	153.500	153.750	11.9375	12.1875	15.1875	15.4375
26	30.6250	30.7500	35.000	152.125	152.500	11.6250	11.8750	15.0625	15.3125
27	30.5625	30.7500	35.000	152.875	153.500	11.6250	11.8750	15.0625	15.2500
28	30.5000	30.6875	35.000	152.875	153.625	11.7500	12.0000	15.1250	15.3750
31	30.2500	30.4375	35.250	152.500	153.375	11.8125	12.0000	15.1250	15.3125
Average for month	31.890	.....	36.190	151.280	.....	11.958	12.170	14.869	15.140

The United States quotations are our appraisal of the major markets for domestic consumption based on sales reported by producers and agencies. They are reduced to the basis of cash, New York or St. Louis, as noted. All prices of domestic class are in cents per pound.

Copper, lead and zinc quotations are based on sales for both prompt and future deliveries; tin quotations are for prompt delivery only.

Quotations for zinc are for ordinary Prime Western brands. Zinc in New York is now quoted at 0.35c. per pound above St. Louis, this being the freight differential. Contract prices for High-Grade zinc delivered in the East and Middle West are 1c. above St. Louis

Silver and Sterling Exchange

	New York		London Spot		Sterling Exchange	
	1931	1932	1931	1932	1931	1932
January	29.423	29.780	13.810	19.623	485.260	342.515
February	26.773	30.136	12.432	19.573	485.551	345.141
March	29.192	29.810	13.524	18.336	485.596	363.463
April	28.279	28.298	13.120	16.923	485.764	374.731
May	27.650	27.755	12.858	16.868	486.188	367.370
June	27.250	27.466	12.707	16.844	486.291	364.471
July	28.255	26.700	13.197	16.930	485.349	354.865
August	27.524	27.986	12.815	18.000	485.284	347.449
September	28.180	27.870	14.101	17.998	451.245	347.003
October	29.538	27.195	17.153	17.813	388.029	339.500
November	32.223	.....	19.393	.....	371.130	.....
December	30.120	.....	20.023	.....	336.798	.....
Year	28.700	.....	14.594	.....	452.707	.....

New York quotations, cents per ounce troy, 999 fine. London, pence per ounce, sterling silver, 925 fine. Sterling exchange in cents.

Copper

	F.O.B. Refinery		London Spot			
	Electrolytic		Standard		Electrolytic	
	1931	1932	1931	1932	1931	1932
January	9.838	7.060	44.938	39.459	47.524	46.200
February	9.724	5.965	45.372	36.917	47.950	41.381
March	9.854	5.763	44.818	33.039	47.699	36.786
April	9.392	5.565	42.694	29.943	45.375	34.190
May	8.665	5.237	38.897	28.548	42.175	32.833
June	8.025	5.145	35.827	26.872	38.966	30.841
July	7.698	5.053	34.402	26.071	37.293	29.107
August	7.292	5.219	32.572	31.401	35.388	34.784
September	6.988	5.978	31.503	34.986	36.148	38.318
October	6.775	5.733	34.957	31.890	41.000	36.190
November	6.558	.....	35.854	.....	41.190	.....
December	6.580	.....	38.273	.....	44.409	.....
Year	8.116	.....	38.342	.....	42.093	.....

New York quotations, cents per pound. London, pounds sterling per long ton. Export copper, f.o.b. refinery: Average for October, 5.256

Lead

	New York		St. Louis		London			
	1931	1932	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos.
January	4.802	3.750	4.604	3.550	13.872	13.905	15.084	15.128
February	4.552	3.712	4.340	3.499	13.444	13.550	14.560	14.571
March	4.527	3.150	4.276	2.993	13.128	13.355	12.345	12.634
April	4.412	3.000	4.164	2.900	12.375	12.606	11.223	11.503
May	3.818	3.000	3.651	2.900	11.491	11.778	10.673	11.036
June	3.917	2.993	3.761	2.896	11.582	11.952	9.608	9.898
July	4.400	2.747	4.225	2.611	12.731	12.899	9.818	10.152
August	4.400	3.235	4.225	3.093	11.944	11.994	11.349	11.588
September	4.400	3.465	4.217	3.315	11.932	12.026	13.122	13.349
October	3.964	3.052	3.773	2.939	13.227	13.270	11.958	12.170
November	3.937	.....	3.758	.....	14.577	14.491	.....	.....
December	3.792	.....	3.592	.....	15.188	15.361	.....	.....
Year	4.243	.....	4.049	.....	12.958	13.099	.....	.....

New York and St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Tin

	New York		London	
	1931	1932	1931	1932
	Straits		Spot	
January	26.137	21.804	115.798	140.219
February	26.315	22.018	117.919	139.143
March	27.065	21.863	121.852	129.810
April	25.222	19.244	112.775	108.935
May	23.221	20.948	104.331	122.286
June	23.478	19.659	104.966	114.530
July	24.978	20.931	111.478	125.863
August	25.738	22.985	114.875	142.017
September	24.618	24.779	117.813	152.705
October	22.723	23.936	126.932	151.280
November	22.779	.....	132.857	.....
December	21.328	.....	138.909	.....
Year	24.467	.....	118.375	.....

New York quotations, cents per pound. London, pounds sterling per long ton.

Zinc

	St. Louis		London			
	1931	1932	1931 Spot	1931 3 Mos.	1932 Spot	1932 3 Mos
January	4.035	3.011	12.747	13.113	14.416	14.834
February	4.012	2.817	12.303	12.694	13.872	14.289
March	4.002	2.787	12.190	12.676	12.616	13.024
April	3.717	2.725	11.353	11.838	11.670	11.958
May	3.306	2.532	10.484	10.875	12.432	12.682
June	3.416	2.777	11.270	11.750	11.548	11.866
July	3.893	2.537	12.280	12.802	11.592	11.967
August	3.817	2.758	11.444	12.028	13.594	13.844
September	3.744	3.322	11.571	12.063	15.455	15.616
October	3.377	3.027	12.733	13.216	14.869	15.140
November	3.209	.....	13.845	14.247	.....	.....
December	3.149	.....	14.361	14.818	.....	.....
Year	3.640	.....	12.215	12.677	.....	.....

St. Louis quotations, cents per pound. London, pounds sterling per long ton.

Cadmium and Aluminum

	Cadmium		Aluminum	
	1931	1932	1931	1932
January	67.115	55.000	23.300	23.300
February	55.000	55.000	23.300	23.300
March	55.000	55.000	23.300	23.300
April	55.000	55.000	23.300	23.300
May	55.000	55.000	23.300	23.300
June	55.000	55.000	23.300	23.300
July	55.000	55.000	23.300	23.300
August	55.000	55.000	23.300	23.300
September	55.000	55.000	23.300	23.300
October	55.000	55.000	23.300	23.300
November	55.000	.....	23.300	.....
December	55.000	.....	23.300	.....
Year	56.010	.....	23.300	.....

Aluminum in cents per pound, 99 per cent grade. Cadmium, cents per pound.

Antimony, Quicksilver, and Platinum

	Antimony (a) New York		Quicksilver (b) New York		Platinum (c) New York	
	1931	1932	1931	1932	1931	1932
January	7.317	5.976	103.000	64.900	36.000	40.000
February	7.069	6.489	100.205	66.304	34.000	40.000
March	7.127	6.188	100.423	72.537	29.538	40.000
April	6.888	5.746	102.077	72.125	26.346	40.000
May	6.524	5.170	101.140	66.380	24.980	39.500
June	6.342	5.034	92.058	59.481	37.115	37.500
July	6.802	5.000	85.808	53.580	40.000	35.200
August	6.596	5.144	80.115	47.444	40.000	35.000
September	6.542	5.606	76.300	47.500	40 000	33.800
October	6.517	5.575	72.385	47.600	40 000	33.000
November	6.679	.....	68.587	.....	40.000	.....
December	6.231	.....	66.115	.....	40.000	.....
Year	6.720	.....	87.351	.....	35.665	.....

(a) Antimony quotations in cents per pound, for ordinary brands. (b) Quicksilver in dollars per flask of 76 lb. (c) Platinum in dollars per ounce troy.

Pig Iron[1]

	Bessemer		Basic		No 2 Foundry	
	1931	1932	1931	1932	1931	1932
January	17.50	16.00	17.00	15.00	17.00	15.50
February	17.29	15.68	16.79	14.68	16.79	15.18
March	17.00	15.50	16.50	14.50	16.50	15.00
April	17.00	15.50	16.50	14.50	16.50	15.00
May	17.00	15.50	16.38	14.50	16.44	15.00
June	17.00	14.85	15.50	14.35	16 00	14.75
July	17.00	14.56	15.50	14.00	16.00	14.50
August	17.00	14.50	15.50	14.00	16.00	14.50
September	17.00	14.50	15.50	14.00	16.00	14.50
October	16.92	14.50	15.42	14.00	16.00	14.50
November	16.50	.....	15.00	.....	16.00	.....
December	16.25	.....	15.00	.....	15.75	.....
Year	16.96	.....	15.88	.....	16.25	.....

Iron in dollars per long ton. [1]F.o.b. Mahoning and Shenango Valley furnaces; freight to Pittsburgh, $1.76.

prices for the Prime Western grade.

Quotations for lead reflect prices obtained for common lead, and do not include grades on which a premium is asked.

London prices for lead and zinc are the official prices for the morning session of the London Metal Exchange; prices for copper and tin are the official closing buyers' prices. All are in pounds sterling per long ton (2,240 lb.).

New York silver quotations are as reported by Handy & Harman and are in cents per troy ounce of silver, 999 fine. London silver quotations are in pence per troy ounce of bar silver, 925 fine. Sterling prices represent forenoon market demand. Cables command a premium.

Current Statistics of Production and Stocks of Copper, Lead, and Zinc

Data from American Bureau of Metal Statistics, American Zinc Institute, and Metallgesellschaft.

All Figures Except Tin Represent Tons of 2,000 Lb.

	1931			1932								
	Oct.	Nov.	Dec.	Jan.	Feb.	March	April	May	June	July	Aug.	Sept.
COPPER — North and South America												
Production, refined, daily average	2,918	2,693	2,780	.....	.....	.....	.....	.....	.....	.....	.....	.....
Domestic shipments	51,348	37,436	36,972	.....	.....	.....	.....	.....	.....	.....	(a) 18,300	(a) 16,700
Foreign shipments	21,030	20,257	26,032	.....	.....	.....	.....	.....	.....	.....	(a) 48,000	(a) 53,700
Stocks, blister and refined	.....	.....	722,014	.....	.....	.....	.....	.....	.....	.....	.....	(b) 775,000
LEAD — United States												
Production, refined, from domestic ore	36,546	31,671	33,576	32,180	28,081	30,345	23,236	25,902	26,068	15,819	17,118	20,498
Production, secondary and foreign	4,722	3,820	4,031	3,554	3,920	3,989	3,134	2,941	2,033	3,754	6,308	2,315
Production, total, daily rate	1,331	1,180	1,213	1,185	1,103	1,107	879	930	936	631	756	760
Shipments reported	34,276	31,216	30,297	27,867	26,319	31,162	26,081	24,258	21,511	19,723	28,248	26,413
Stocks, end of month	139,796	144,057	151,380	160,257	165,933	169,091	169,091	173,929	180,460	180,255	175,426	171,831
ZINC — United States												
Production, daily average	699	684	709	723	742	726	688	601	547	476	432	434
Domestic shipments	21,163	20,327	21,005	22,472	21,896	22,576	18,046	18,087	14,958	12,896	18,069	21,132
Export shipments	.....	.....	.....	31	.....	.....	.....	.....	24	.....	39	20
Stocks, end of month	130,666	130,865	129,825	129,886	129,506	129,451	132,025	132,580	134,032	135,907	131,203	123,056
World Production Rate (Daily Average)												
Copper	3,940	4,070	3,940	.....	.....	.....	.....	.....	.....	.....	.....	(b) 2,500
Lead	3,806	3,980	3,976	3,938	3,872	3,598	3,587	3,711	3,497	3,190	3,334	3,303
Zinc	2,736	2,725	2,601	2,600	2,615	2,623	2,528	2,392	2,343	2,224	2,121	2,182
Tin (Metric Tons)	352	325	351	311	274	340	312	311	261	178	174	.....

(a) Deliveries; private estimate. (b) *E.&M.J.* estimate.

Editor: A. W. ALLEN

Associate Editors: A. H. HUBBELL, H. D. KEISER

Assistant Editor: W. N. P. REED

Market Editor: H. H. WANDERS

Publishing Director: H. W. CLARKE

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